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MINE SAMPLING

AND

VALUING

A Discussion of the Methods Used in Sampling and Valuing Ore Deposits with Especial Reference to the Work of Valuation by the Independent Engineer

BY

C. S. HERZIG

WITH A CHAPTER

ON

SAMPLING PLACER DEPOSITS

BY

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UNIV. OF CALIFORNIA

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This Book is Dedicated to

T. A. RICKARD

"He Blazed the Trail"

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CONTENTS.

Part II.

MINE VALUING.

Chapter XI.

Chapter XII.

Chapter XIII.

Chapter XIV.

Chapter XV.

Chapter XVI.

Chapter XVII.

Chapter XVIII.

Chapter XIX.

Chapter XX.

INTRODUCTION.

In the past a great deal has been written on the subject of mine sampling and valuation, but for the most part the literature is fragmentary, and I believe that scarcely anywhere is there any reference made to its elementary side. Somehow or other, engineers pick up a knowledge of this part of the business, although I do not think that the methods of mine sampling are taught in schools of mines, nor do the text-books on mining deal with it in detail.

A few years ago T. A. Rickard, who at that time was editor of the *Engineering and Mining Journal* of New York, wrote a series of articles that were followed by contributions from various engineers. These articles and the ensuing discussion were eventually published in book form with the title of 'The Sampling and Estimation of Ore in a Mine.' Aside from this, no attempt at a comprehensive work on the subject exists, and for that reason Mr. Rickard's book has since then served as a sort of text-book to the younger engineers desiring information on this phase of mining engineering. The discussion contained therein was a most valuable one at the time, but on the whole it lacks that continuity of thought which the subject deserves.

We owe, too, a great deal to H. C. Hoover, who, in his book 'Principles of Mining,' has contributed a valuable discussion on mine valuation. It is a source of wonder to me that this matter has not been seriously taken in hand heretofore, because it must be remembered that the whole success of mining operations is often absolutely dependent on accurate mine sampling. I therefore trust that the following pages may serve as a guide to the young engineer and may not be without value to the more experienced and older men.

I am indebted to Messrs. Frank H. Probert and Lloyd T. Buell for information regarding churn drilling, as also to E. N. Skinner; to H. S. Munroe for his views on the gravimetric method of calculating averages and to others who have been good enough to offer suggestions of various kinds.

C. S. HERZIG.

London, December, 1913.

PART I.
MINE SAMPLING.

UNIV. OF CALIFORNIA

CHAPTER I.

PRINCIPLES UNDERLYING MINE SAMPLING AND VALUING.

Briefly stated, the object of mine sampling is to determine the grade of the ore exposed in a mine, and thereby lead to an estimation of the assay-value and tonnage of the orebody. Mine valuation, on the other hand, is a broader subject, comprising many ramifications, and requires technical skill and experience, as well as business ability.

The principles underlying the sampling of mineral deposits are the same whatsoever the kind of mineral, be it gold, silver, copper, lead, tin, iron, or any other. Variations in the methods employed depend on the intrinsic or commercial value of the mineral, its regularity of distribution in the gangue and the idiosyncrasies of the engineer. The object to be attained is to estimate as nearly as possible the economic value of the ore, keeping in view the cost of the examination.

The estimation of the tonnage and assay-value of an ore deposit is an attempt to put into exact figures the result of inexact work. With all the care possible in the sampling operations, the samples taken represent only the outer skin of a block of ground and the next face may be 100 ft. distant. Given an orebody exposed by certain workings, the problem is to determine the numbers of tons of material that can be mined from it and what will be its mineral or metallic contents when so mined. The adjustment to meet the condition of profit as dependent on metallurgical or economic conditions is a subsequent operation that calls for wide experience and especial talent on the part of the engineer. Not all engineers of experience are good mine valuers, any more than all experts with oils are painters.

A successful mine valuer requires an unusual combination of qualities. He needs to be experienced, observant, analytical, constructive, bold yet cautious, far-seeing and have the commercial instinct strongly developed. The commercial instinct is the one quality the engineer most rarely possesses and the lack of it is the cause of many disasters. It must be borne in mind that undervaluation is just as bad engineering as overvaluation. The former is largely due to timidity or excessive caution and the latter to excessive optimism. The engineer's work should be constructive. The over-cautious mine valuer is destructive; he not only destroys business, but often unjustly destroys the fruits of other men's labor. In doing so he violates his function as an engineer.

Psychology plays no unimportant part. A man's first impression is apt to affect his final judgment. The wish is often father to the thought—physical discomfort or other causes may instil a restlessness or perhaps a hope that a preliminary examination may give sufficient information for an unfavorable report. We can reason out any desired conclusion, if we but assume a suitable hypothesis. An examining engineer must keep an open mind and must not form a premature judgment based on insufficient data. He must be a mere collector of facts until it is time to form a decision. An assay plan has changed many preconceived notions. The facts must be collected and put away in the proper pigeon-hole of one's mind until required, for if the engineer is biased, either consciously or unconsciously, such bias will be a factor in forming the ultimate opinion.

One often hears the question asked whether a report has been made for a seller or a buyer, indicating a possibility that the engineer's judgment may be affected by the character of his employment. I wish to make a strong plea for a standard of action among engineers, for a mode of procedure such as is customary, for instance, in the medical profession. An engineer making a mine examination should view his work in the same light that a doctor does a patient and should make a diagnosis, regardless of any outside consideration. As engineers we cannot provide against the moral turpitude of individuals, but at least we can adopt a standard of action that conscientious men may follow, so as to destroy the evil effects of mining charlatans and adventurers, who happily do not flourish so profusely now as formerly, and thus raise the profession in the eyes of the general public to the position it deserves.

Before proceeding further, it is advisable to define the word 'ore' as it will be used in this discussion and as the author believes it is used by most mining engineers. J. F. Kemp, professor of geology at Columbia University, in a paper presented to the Canadian Mining Institute,[1] after quoting most of the definitions of ore then extant and discussing them, defined ore as follows:

"In its technical sense an ore is a metalliferous mineral or an aggregation of metalliferous minerals, more or less mixed with gangue and capable of being, from the standpoint of the miner, won at a profit, or from the standpoint of the metallurgist, treated at a profit."

This definition seems to be too restrictive when judged from the ordinary mining standpoint. Personally, I prefer either of the following conceptions, the first one being a modification of that appearing in Murray's 'New English Dictionary' (1908), and the second by R. H. Stretch.

(1). "A native mineral *or aggregation of minerals* containing *one or more* precious or useful metals in such quantity and in such chemical combination as to make its extraction profitable." (The italics show the changes from Murray's definition.)

(2). "An 'ore,' strictly speaking, is a single mineral which is a chemical compound of a useful metal and some other element

[1]*Mining and Scientific Press,* vol. 98; p. 419.

or acid. In common usage, however, complex mixtures of pure minerals are considered as single ores; while free gold, native silver, and native copper, together with their accompanying gangue minerals, are also classed as ore. Among miners whatever will pay to treat or ship and sell, is considered ore, *as also low-grade mineral, which might be utilized by concentration or improved facilities; but there is an indefinite shading off into material containing traces of ore-minerals but hopelessly unavailable, and this is not considered ore;* neither are gold gravel or platinum sand called ore."

Stretch emphasizes the shading off of value indicating the general usage of mining men, that not until the percentage of valuable mineral is practically negligible is the term dropped.

The generally accepted definition among both technically trained and non-technical mining men is that it is used to designate a metallic mineral or minerals occurring in such quantity which we know from experience to be of commercial value. In other words, the miner (i. e. one who mines) is continually discounting the improvements in science and by common accord designates as ore, not only material which can be made to yield a profit at the moment, but also such as contains appreciable quantities of the valuable mineral being exploited, as distinguished from the purely barren gangue and the country rock.

In discussing Kemp's paper, I suggested the following definition in place of the one offered by him:

"An ore is a metalliferous mineral or an aggregation of metalliferous minerals, more or less mixed with gangue, that from the standpoint of the metallurgist can be treated at a profit, or that from the standpoint of the miner occurs in a vein, lode or other geological deposit and concentrated by nature in such a manner as to attract the attention of the miner on account of his belief that he can work at least a portion of such deposit at a profit."[2]

I am in agreement with Kemp so far as the metallurgist's viewpoint is concerned, but differ in other respects. I believe however that my definition conforms to the view most usually held by mining men.

Many engineers' reports nowadays state not only the tonnage of ore amenable to profitable treatment under the conditions existing at the time of their examination, but likewise give the tonnage and value of 'low-grade ore' that experience teaches may reasonably be expected to lend itself to profitable handling at a later date.

All mining operations are conducted with a view to earning a profit, hence that question cannot be eliminated, neither does it limit the use of the word among the men whose use makes or unmakes its meaning. No man starts a business of any kind unless he expects to make a profit, but not all business is done at a profit In attempting to apply the standard of profit to the use of the word 'ore,' its advocates are placing a restrictive meaning on it

[2]*Mining and Scientific Press,* vol. 99; p. 117.

that is neither understood nor applied by the professional man nor yet the practical miner. For instance, Rickard defines ore as metal-bearing rock, which at a given time and place can be exploited at a profit.

Mine No. 1 may be making a profit, while its neighbor working the same ore makes a loss, due to bad management or inadequate plant. It is absurd to designate one as ore and the other as waste. Because the business is run at a loss does not deprive it of its commercial aspect. Just as in any other branch of commerce, the idea of profit is present in all mining operations and is the chief incentive, but failure to attain the sought-for end does not change the character or name of the commodity. So with the word 'ore,' if by the time the metal produced is sold in the market and a loss is shown in the operations, the metal-bearing rock from which it was derived is none the less 'ore.'

Let us consider another aspect of the case. No doubt "use is the law of language" and except for the small circle who are attempting to restrict the meaning to the standard of profit, the meaning of the word 'ore' is pretty generally understood throughout the mining world. If one speaks of ore and knows the mine, there is no possibility of misunderstanding what is meant. If the word 'ore' is restricted to metal-bearing rock that can be exploited at a profit and everything else is waste, then we are confronted with the necessity of coining another expression for the intermediate product between ore and the absolutely barren country rock or barren gangue material.

To give a concrete case, assume an orebody, consisting of three distinct kinds of material, namely:

1. Ore that will yield a profit.
2. Ore which is too low grade to yield a profit.
3. Absolutely barren quartz, or other similar worthless gangue.

Where these materials occur in distinct bunches, shoots, zones, bands or other manner, mining operations will be carried on with a view to breaking only class No. 1, namely, the ore that will yield a profit. Class No. 2 may with better economic conditions, such as improved metal prices, cheaper supplies, etc., become profitable at any time.

No mining man would use the same expression to denote Class 2 and Class 3 and would have no hesitancy in speaking of barren gangue and country rock as waste at any time, regardless of the economic conditions, or metal prices. No one would hesitate to use rock of this sort as filling or for road metal, or for any other similar purpose, where it is beyond recovery. On the other hand, the materials comprising Class 2 would be carefully guarded and certainly would not be called waste. If it is not ore, then another name must be given to it, which we would have to learn. With the present use of the expressions 'profitable ore' and 'unprofitable ore' there can be no misinterpretation and certainly these expressions are as suitable as any new ones that may be coined.

Therefore, the sampling of a mineral deposit must be conducted with a view not only of determining the tonnage and value of the *profitable ore,* but the tonnage and value of low-grade or *unprofitable* ore as well, so far as the conditions justify the necessary expenditure. The miner is first of all concerned with the gross value of the metallic content. Fluctuating prices of metals, or efficiency of metallurgical treatment, which may govern the question of profit, do not restrict his use of the word *ore.*

Mining engineers are most particularly occupied with the exploitation of rocky masses occurring as veins or orebodies surrounded by rock usually of a different character, in any case so far different as to be differentiated as waste rock. From the very nature of their occurrence these orebodies usually extend to a relatively considerable depth below the surface of the ground. Therein are presented some of the inherent difficulties attendant on the work. Underground workings are required for their exploitation which are usually dimly illuminated, wet, cramped, and obstructed by timber. The following pages, therefore, deal with the sampling and estimation of mineral deposits *in situ* opened by underground methods; not excluding, however, low-grade copper, iron, or similar deposits that for economic reasons are being mined by steam-shovels or other mechanical appliances and open to daylight.

In a consideration of the subject of mine sampling, it must be borne in mind that the sampling of a mine by an independent engineer for valuation purposes is a different problem than the periodical estimation of ore by the mine management for reports to directors and shareholders. The latter involves no money transaction and the same care is not required nor is the responsibility so great. The independent engineer uses the utmost care and sacrifices all other things to accuracy. In the daily sampling of a mine the first concern of the management must necessarily be utility and expediency; as a consequence, the daily mine samples are individually apt to vary considerably from the true value. In practice it is usual to apply an empirical factor of correction as giving sufficiently accurate results. The present discussion is more particularly concerned with the aspects of mine sampling for valuation purposes, because on operating mines the management has time to work out the proper factor to meet the existing conditions. On the other hand, the visiting engineer must, in the short space of time occupied by his examination, use methods that will ensure the desired accuracy.

CHAPTER II.

PICK ANALYSIS.

It is bad practice to commence sampling operations before a knowledge of the deposit has been obtained by means of a study of its mineralogical and geological character, both on surface and underground, and this work is what I have termed 'pick analysis.'

It is a curious fact that in the entire literature on the subject of mine sampling and valuation, no attention has ever been called to this work, nor does it seem to be the usual custom of most examining engineers. The general method of procedure with many engineers is to take a few scattered samples as a preliminary step in order to locate the occurrence of values. This method can only be described as haphazard, and is not in line with the systematic methods desirable in such important work as mine examination. Even with the simplest kind of deposit, a systematic investigation should be made to determine its mineralogical and geological characteristics. In other words, we bring into the realm of mining engineering some of the methods employed by the geologist, but with a different object. Taking a few preliminary samples is not a sufficient guide for conducting comprehensive sampling operations intelligently, nor is the more customary practice of merely walking through the mine enough.

Very often the sampling of a mine is started immediately, before any detailed knowledge of the deposit has been acquired, and this is akin to a man walking blindfolded. The ore exposures underground are usually besmudged by dirt, water, and powder smoke, which makes them difficult to examine, and thereore it is necessary to obtain a clean surface, or by chipping off a piece of the face get a fresh fracture, free from surface oxidation. Usually the mine foreman has a fairly accurate idea of the mode of ore occurrence and distribution of values, for the simple reason that he watches the development work from day to day, sees the faces after each round of shots, and thereby is enabled not only to study the geological structure but the mineralogical character of the orebody as well. Mine foremen are scarcely expert geologists, but observation teaches them that the occurrence of valuable minerals in any particular deposit is usually accompanied by certain specific conditions or indications.

The examining engineer, in order to properly sample the mine and afterwards to interpret intelligently the results, must familiarize himself with these facts. He must rapidly acquire the knowledge that it has taken the mine foreman months or years to learn, and it must be done by optical inspection.

As a preliminary, the available mine maps should be studied and the mine officials thoroughly interrogated regarding all questions of interest. In this way a fairly exact knowledge of the conditions may often be acquired before going underground. The ore and waste dumps should be examined to familiarize oneself in daylight with the character of the ore and rock found in the workings. This done, the engineer should, in company with one of the mine officials, make a cursory inspection underground, to enable him to get his bearings. Henceforward he should continue the investigation without the presence of any person other than those in his employ.

Before sampling is attempted, the detailed optical inspection, or pick analysis, of the exposed faces in the workings should be made by chipping the ore and rock exposures at short intervals in the drifts, cross-cuts, and other workings, by means of a prospecting pick or a miner's pick, if the ground be very hard, until a thorough familiarity with the characteristics of the orebody is acquired. The country rock and dikes should receive attention, but the same thoroughness is not required with them except where necessary to solve some geological question. By this means there will be determined the general conditions as to character of mineralization and the amount of sulphides, base metals or associated minerals present, the width of the orebody, whether regular or irregular, the existence of lenses or shoots, the presence of faults and dikes and their effect on the position of the orebody and in some cases their influence on the grade of the ore. This inspection should determine the portions of the orebody to be sampled and those that can be regarded as worthless; in other words, determine the limits of the mineralized zone. With this knowledge in hand intelligent sampling operations can be undertaken.

The usefulness of pick analysis can be better appreciated by citing a concrete case. A few years ago a copper mine, held under option at a high price, was examined. The property was at a considerable distance from a railroad, although a connecting line was contemplated, and the district was one where labor was scarce and costly. The previous history of the mine was that a considerable quantity of ore had been mined and smelted on the spot. Two assistants chipped the faces of the hard rock for my inspection, so that I was enabled by this means to gather sufficient evidence in two days to warrant an adverse report.

By optical inspection a close approximation of the average copper content was made, to check the figures given in the reports, and a simple calculation demonstrated that even allowing all the content in the ore claimed for it by the vendors, there was not enough tonnage available to warrant the price. The mine was at a different time examined by another engineer who spent three or four weeks sampling it, to finally show something like 15,000 tons of 3 to 4% ore in sight. So he turned it down. He might have been saved considerable time and expense by having followed the other method. Pick analysis will often determine the fact that in the bottom level of the mine the valuable minerals are 'petering out.' By first sampling the bottom level, sufficient evidence may be gained to obviate the necessity of any other work. No one wants to buy a mine with a bad bottom, as a diminution of value generally indicates that the limits of the orebody are being approached. Pick analysis will almost always indicate the existence of high-grade streaks and their mode of occurrence, but above all else, it gives the engineer a thorough knowledge of the mode of ore occurrence, character of mineralization, etc., so essential in conducting intelligent sampling operations. This work should never be delegated to subordinates, but must be done by the responsible engineer himself.

In the case of a gold-bearing quartz, with little or no associated sulphide minerals, the information obtained is not so comprehensive, as it is in the case where sulphides occur in appreciable quantity, or in the case of base metal mines, especially those of copper, lead and zinc, where often with a little experience the assay value may be closely approximated by mere optical inspection.

Be the ore what it may and whatever its genesis or type, a thorough investigation by pick analysis will greatly facilitate not only the subsequent sampling operations but will be of value in the ultimate work of estimating the ore reserves. It is usually a tedious job, but one that will well repay the effort.

CHAPTER III.

IMPLEMENTS EMPLOYED IN SAMPLING.

A sampling outfit must consist not only of tools for breaking the rock, but also means to safeguard samples from interference after they have been taken. Certain additional paraphernalia are required, as set forth in the accompanying list, which is recommended as a result of my own experience.

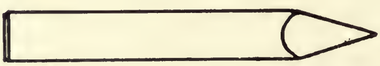
List of Implements.

Hammers.
Moils and (or) gads, and (or) chisels.
Miner's pick.
Prospecting pick.
Shovel.
Whisk-broom, scrub-brush or similar article.
Steel tape 50 or 100 ft.
Steel tape 10 ft.
Canvas sheets.
Duck sampling buckets.
Leather sack with Yale lock.
Small sample sacks.
Twine.
Seal and sealing wax.
Linen baggage labels.
Pencils.
Plumb bobs and lines.
Pocket compass (preferably Brunton).
Whitewash.
Jones sampler or other mechanical divider.
Paper sample envelopes.

HAMMERS.—Cutting a sample is different from cutting a hitch for timber and therefore the hammer should not be of excessive weight. As the force of the blow is regulated by the work to be done, the hammer should be thoroughly under the control of the user. In the machine shop there is a parallel case; the machinist's hammer for cutting steel and iron rarely weighing more than two pounds. Using a four-pound hammer underground after a period of inactivity soon wearies the muscles and lessens control. This causes the operator to miss the head of the tool and means bruised and skinned hands. My own practice is to use an ordinary double-head, single-hand hammer with a broad face and weighing 3 to 3½ lb. for cutting the sample, except where the ground is extremely hard and a double-hand hammer becomes necessary. The larger hammers should weigh about 7 lb. and are otherwise used for breaking rock.

MOILS, GADS and CHISELS—Should be of convenient size and weight. This is regulated by the diameter of the steel used and the length of the tool. Inexperienced people often make the mistake of employing heavy steel. It must be sufficiently stout to withstand the hardness of the ground, but as light as possible commensurate with the work to be done. Heavy steel has the double disadvantage of excessive weight and so large cross-section as to absorb too much of the force of the blow. The most useful size is ¾-in. octagonal drill steel, although ⅝-in. steel may be used occasionally, where circumstances permit.

Weight is an important factor, as often the sampler is in a cramped position, standing on a scaffold, box, saw-horse, or other unstable object with muscles tensed, having to support the entire weight of the steel in his hand, watch the sample, and leave room for his assistant holding the bucket. Under such circumstances a heavy tool may cause so much discomfort as to seriously retard the sampling operations. Moiling is tedious work and exactitude is required, therefore the steel should weigh as little as consistent with the duty expected of it, thus leaving the sampler free to put all his force into the hammer blow.



IMPROPER WAY OF SHARPENING.

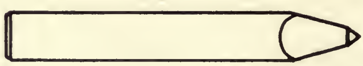


Fig. 1. PROPER WAY OF SHARPENING.

The moil is more generally employed than the gad or chisel, as the sharp point will usually get a bite in the rock quicker than the straight edge of the gad. In sharpening a moil, instead of drawing the point down from the shank of the steel in a simple pyramid, the so-called diamond point is preferable, that is, the extreme point should be a small flat pyramid terminating the larger one, thereby strengthening the tool where it is most needed by presenting a greater cross section of metal. One of the greatest difficulties in moiling rock is the frequency with which the point breaks off. By this method of sharpening, the moil is strengthened and less breakage occurs. See Fig. 1.

The gad or wedge is an ancient tool. It is fitted with a handle and is one of the two tools comprising the emblem known as the crossed hammers. It is primarily intended for breaking out rock by the force of the wedge.

The chisel may often be usefully employed, especially in sampling slate or schist. The tool for sampling is sharpened just like the ordinary machinist's cold chisel, and in slate or schist has the advantage of cutting a more regular channel with the same effort.

Care should be exercised in tempering these tools, the best color being a light pigeon blue or slightly darker. This affords the requisite toughness for hard rock, and at the same time avoids the brittleness of a harder temper. If the ground be extremely hard, better results can be obtained by tempering in oil.

A variety of lengths should be supplied and in hard ground twenty to thirty moils per day may be needed. The most useful length, however, is 10 to 12 inches. Where the roof or face is somewhat out of reach of the sampler, longer steel can be usefully employed, but experience teaches that the shorter moil tends to greater accuracy. It permits close inspection of the cut while the sample is being taken, gives a more direct application of the hammer blow, and, as previously mentioned, the light weight is an important factor to the ordinary engineer not accustomed to manual labor.

PICK and SHOVEL.—These may be of the ordinary type used about the mine, although the poll pick is preferable. The prospector's pick may weigh 1½ to 2 pounds.

STEEL TAPES.—The long tape is used for marking out the sampling intervals and such other measurements as are necessary in the work, apart from surveying. A Chesterman or other first-class 50 or 100-ft. steel tape, divided into feet and tenths, should be used, and in no instance should a linen tape be employed in sampling operations. The short tape should be of the spring type, so as to permit the sampler to extend or shorten it with one hand. This is a small point but may be the means of saving considerable time and annoyance, as a reel tape requires the use of both hands and occasionally when standing on a scaffold or shaky box, holding a candlestick at the same time, even this small operation may become extremely irksome.

CANVAS CLOTH.—A canvas sheet 5 or 6 feet square is required as a mixing cloth, as well as several pieces 3 ft. square. Enquiries made in England a few years ago show that canvas 6 ft. in width is not manufactured in England, although it is easily obtainable in the United States. Such being the case, engineers in the former country will of necessity have to have two widths sewn together.

DUCK SAMPLING BUCKET.—For fifteen years I have used a duck or canvas bucket in preference to a box or other receptacle or to the large canvas sheet so often employed for collecting the chips of ore broken from the face to form the sample. In my opinion these buckets have many advantages over any other thing that is ordinarily used for this purpose. The unconscious salting of a sample where brittle sulphides occur is minimized, and the spoiling of a sample by a fall of ground, so common in sampling work, is practically eliminated.

One of these buckets weighs but a few ounces as compared with the considerable weight of a wooden box. The strain on one's muscles in holding a candle box in the position shown in the frontispiece of T. A. Rickard's book on the 'Sampling and Estimation of Ore in a

Mine' can easily be recalled by anyone who has tried it. There are many places in a mine where the receptacle must be held at arm's length overhead, and the mere position is sufficiently tiring without sustaining anything heavy. When the weight of the sample is added to the already heavy box, the work becomes irksome indeed, and a slight shifting of the box, as the assistant's muscles get tired, may cause the sampler to bash his knuckles and visit his wrath upon the head of the poor fellow holding the box. As compared with this, the canvas bucket not only has the advantage of greater lightness, but in view of its flexibility, permits the assistant to more intelligently follow the operations of the sampler and thereby facilitate the work. Should the sampler miss hitting the moil, it is infinitely less aggravating to strike a duck bucket with one's knuckles than the sharp edge of a candle box.

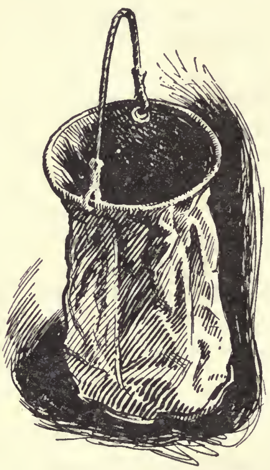


Fig. 2. CANVAS SAMPLING BUCKET.

Fig. 3. A BUCKET THROTTLED FOR SOFT GROUND.

In soft ground where the ore is apt to come away in chunks, the bucket can be throttled in the middle by the hand, so as to form two compartments and if an excessive amount does come away the excess can be rejected before the remainder is added to the sample. With a candle box under the same circumstances, either the sample would have to be discarded entirely, or an inaccurate sample taken. See Fig. 2.

The bucket is about 9 in. diameter and 14 in. deep. Suitable buckets can often be purchased from camp outfitters. If necessary they can be made in any small town where duck or canvas is obtainable. Any blacksmith can make the ring, which should be of $^3/_{16}$-in. iron, over which the duck is sewed in such a manner that all the seams are outside. The long seam down the side should be a lap seam similar to the seam in a sail, without any rough edge, the selvedge if possible being on the inside of the bucket. The bottom should be put in with the seam outside, the rough edges bound with tape or thin, pliable leather to give a neater appearance and to prevent the unraveling of the fabric. A handle of ¼-in. rope will be found a convenience. See Fig. 3.

The ring forming the mouth of the bucket is sometimes made of copper wire. The ends can be joined by brazing a sleeve over them

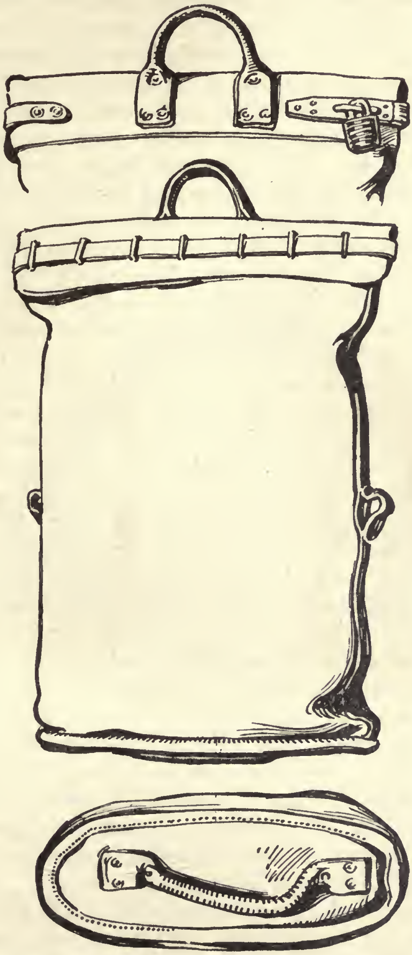


Fig. 4. LEATHER SACK FOR CARRYING SAMPLES.

or they can be simply doubled over themselves. The use of the wire permits shaping the ring by hand in such a manner as to allow the bucket to be held closer to the face. In my opinion this arrangement is of doubtful advantage. This wire need not be of such large diameter as the iron.

The LEATHER SACK.—A leather sack (see Fig. 4), about 30 by 18 in. or 26 by 15 in., in the form of a United States mail sack and fitted with a Yale lock, should be carried for safeguarding samples both underground and on the way from the workings until they can be placed securely under lock and key on the surface. The use of such a leather sack is almost a sure preventive against salting at this period of the operations. Salting to be well done necessitates that the salter sees what he is doing, so that he can increase the gold contents of each individual sample by a desired amount.

Once locked up in the mail sack, the only way of getting at the samples to salt them is by making an incision in the leather and injecting a solution into the sample. As only a portion of the total number of samples can be reached in this way, proper salting is interfered with, and as the salter is working in the dark such erratic assay results are likely to be obtained as to lead to immediate investigation and detection. An incision made in the leather may escape notice, but the imposture would be discovered by the engineer as soon as he arrived on the surface, where it should be his immediate duty to take the samples from the mail sack and place them in safety in the assay office, box, or whatever may be his method of safeguarding them until ready for assaying. While making this transfer of samples, the excessive wetness of the sacks, where the solution has been injected, will generally be readily apparent, even if there were no means of discovering the salting during the subsequent operations.

In the United States these sacks can be purchased ready-made from leather goods manufacturers in various parts of the country; in England they can be made by any good saddler. The sack should be made of stout leather, the side seam secured by copper rivets spaced close together so as to make the joint secure. The bottom should be riveted to the sides with a lap joint, the overlapping ends being bound with leather.

There should be a handle both top and bottom and two loops on either side big enough to allow a rope to be passed through them, so that the sack may be securely fastened to a pack saddle. Very often these sacks are the only means the engineer has for carrying his mine samples in mountainous districts and the loops are of great assistance when pack animals are the only means of transport.

SAMPLE SACKS.—Two sizes of sample sacks are ordinarily used, the smaller 6 by 12 in., and the larger 9 by 14 in. They should be made of duck, or light-weight canvas, double sewn with strong linen thread, and should always be kept under lock and key to prevent tampering. When this formality has been neglected, the sacks are turned inside out and thoroughly beaten before use. When in use the seams should be inside.

SUNDRY ARTICLES.—Abundant sealing wax of first-class quality should be provided and a well cut and distinctive seal about ¾ or 1 in. diameter used to seal the sacks. Lead seals are sometimes used instead of wax, but they are cumbersome and give no better result. Besides, sealing wax can be purchased almost anywhere.

Linen baggage labels of a small size are the best for marking samples, as rough handling does not destroy them. Slips of paper put in wet ore or coarse, sharp, hard ore are apt to become mutilated and illegible.

An indelible pencil should be used for marking the sample numbers. A hand compass is a necessity underground for taking bearings. Plumb-lines form a useful adjunct. Whitewash can usually be obtained at the mine, but even where this is not available, white or colored chalk or lumber crayon should be used to indicate the sampling intervals in the workings.

For tying the sample sacks, fairly strong twine should be used. I find it a great convenience, before going underground, to cut this twine into lengths of about 10 or 12 in. and secure 40 or 50 pieces together by an elastic band, doubled over several times and rolled down to the middle in such a manner that one piece of twine can be pulled out without disturbing the others. Fig. 5. Some engineers go in for a distinctively colored twine, but so long as it is strong

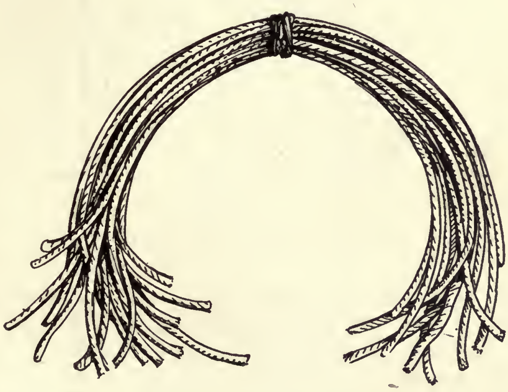


Fig. 5. TWINE CUT IN LENGTHS READY FOR USE.

the color matters not one iota. If the bags are properly sealed, the string cannot be molested without deteection.

Where the sample is quartered instead of divided over a mechanical divider, a brush of some kind is essential to remove the fine ore remaining on the canvas and belonging to the quarters that are removed. A brush is also essential for gathering the fine ore composing the final portion of the sample, where the valuable metals are contained in brittle sulphides, as the loss of the fine ore may materially affect the assay value of the whole sample. It is also used for cleaning the canvas sheets and buckets after the completion of each sample.

MECHANICAL DIVIDERS.—There are various forms of mechanical dividers or samplers in use, which can be purchased from dealers in assay supplies. Such apparatus is not only a great time saver, but gives more accurate results than the ancient method of quartering, which is still used to a great extent under suitable conditions. The most useful form for mine examination work is the so-called Jones sampler. This divides the sample in halves, both portions being caught in separate receptacles placed underneath the spouts. (Fig. 6).

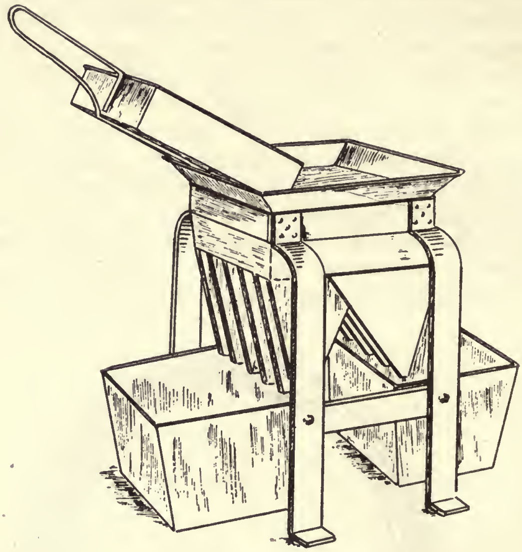


Fig. 6. JONES SAMPLER.

The theory of the divider is that if the ore be made to flow in an even stream over the top of the apparatus, it is automatically divided by means of equidistant partitions, which intercept the stream of ore. The quantities passing respectively through alternate sections combine so that the sample is divided into two equal parts, theoretically having the same assay value.

If the Jones sampler is not obtainable, a simpler form, but one giving equally accurate results, is the so-called riffle divider (Fig. 7),

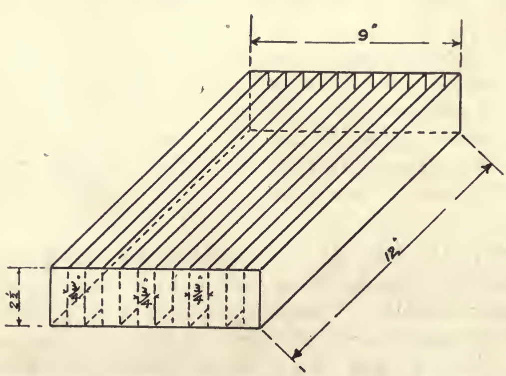


Fig. 7. RIFFLE SAMPLER OR DIVIDER.

which can be made by any tinsmith. This is a rectangular affair, a useful size being 12 inches long by 9 or 10 in. wide and about 2½ in. deep. The partitions may be ⅝ to ¾ in. apart, every other one being a trough followed by a space. This is set up in any convenient way and the division of the sample is made by rejecting the portion passing through and retaining the portion remaining in the troughs. On an important examination in Burma a few years ago, such a home-made divider was used successfully in the preparation of about 500 large samples, some of which weighed as much as 200 pounds.

CHAPTER IV.

GENERAL METHODS OF SAMPLING.

T. A. Rickard has pointed out [1] that with an ore of the consistency of cheese an accurate sample can be taken by running a scraper over it, so as to make a narrow furrow across the full width of the ore. Unfortunately the operation cannot usually be accomplished so simply. Nevertheless, be the ore hard or soft, tough or brittle, compact or open in texture, the general principle underlying the work is the same. What is required is a true cross-section.

The method of taking a sample as ordinarily described in the literature on the subject, is to cut a channel so many inches wide and so many inches deep across the orebody. At other times it is advised that a symmetrical V-shaped groove should be cut through the ore. In my opinion it is practically impossible to secure an accurate result by cutting a channel of a given depth across the ore, for the reason that no matter how carefully, within practical limits, the face is trimmed, there are so many irregularities in the face, that unless an abnormally large sample is taken, such a channel requires taking disproportionate quantities of certain parts of the exposure. As reiterated throughout this discussion, the sampler must use judgment. *Equal quantities as measured by the eye are to be taken.*

The general method of taking a sample can best be illustrated by describing a typical case, such as a vein of white quartz, between hard and solid smooth walls of dark-colored rock. Here is an orebody that to all intents and purposes can be mined clean without the admixture of any country rock, therefore a sample properly taken between the walls will represent the value of the ore as mined.

The place where the sample is to be taken is indicated by marking the walls with whitewash slightly colored if necessary, colored chalk or candle smoke if nothing else is available. The sample must be taken from wall to wall opposite this point, along a line at right angles to the dip of the orebody, care being used to secure proportional amounts throughout the cut, whatsoever the texture or condition of the rock. The groove is preferably cut by means of a moil and hammer and the chippings caught in a sampling bucket, candle box or other receptacle, and on completion of the sample the rock chippings are put carefully into a sample sack, together with a label marked with the proper sample number, after which the sack is securely tied up and sealed. The bag containing the sample should be immediately placed in the leather mail sack, and if this is not always left within sight of the sampler, it should be secured by a lock to prevent any possibility of tampering.

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[1] 'The Sampling and Estimation of Ore in a Mine,' page 20.

This in briefest outline, is the method to be pursued, although there are a great many details connected with these various operations on which not all engineers are in agreement and which on account of their difficulty of application, are often slighted in actual practice by careless or lazy samplers. These points will be discussed in the subsequent pages.

Preparation of Exposure.—The engineer is called upon to sample rock faces of all kinds from surface exposures overgrown by vegetation, more or less covered with earth or fungus, to caving ground below surface protected by heavy timber. Aside from the physical obstructions, blasting operations may leave the face to be sampled of irregular shape. In theory, the width sampled, namely, the perpendicular distance between the walls at right angles to the dip, should be represented by a rock face of that length and no more. In practice this condition is rarely met with.

Before sampling is commenced, the face to be sampled must be thoroughly cleaned and squared-up. Any vegetable matter or other foreign substances, such as soil or mud, must be removed, all irregularities of surface trimmed off, arched corners broken down and in general the exposure so trimmed that it conforms as nearly as possible to the theoretical requirement. In the very nature of things this requirement is ordinarily impossible of attainment in practice. I do not mean by this that it is unnecesary to square-up or to clean a face to be sampled. On the contrary, I am most insistent that this should be done in all cases, so far as may be practicable, yet under the guise of impracticability the sampler should not take refuge to do slovenly, and, therefore, inaccurate sampling. Occasionally it may be possible to use explosives to assist in the necessary preparation, but in the majority of cases only pick and hammer are permissible. With all the care possible, the face presented after this work is completed, as a rule will be full of inequalities and irregularities, nevertheless the engineer must conform to conditions as he finds them. Sampling ore is not as easy as sampling cheese, and despite any irregularities of surface a representative sample must be taken.

In addition to the precautions already mentioned, it is most desirable, especially in damp or wet mines, to pick down part of the ore, to present a clean face for sampling. In all mines the rock faces become more or less covered with powder smoke and dirt; this should be removed before sampling. In dry mines it can often be accomplished by merely cleaning the face with a stiff brush such as a scrub brush or wire brush. This film of dirt, if permitted to remain, not only prevents a satisfactory inspection of the face to be sampled, an important adjunct to proper sampling, but may vitiate the result as well. Another reason for picking down the face is to diminish the likelihood of salting, should it have been prepared for that purpose. As samples are usually taken at regular intervals, interested parties may measure ahead and inject gold into the rock crevices.

Where it is possible to square up and clean a face it only remains to take the sample. Many times, however, it is impracticable to square a face to conform to the rule. Not only is this true of wide orebodies but of narrow ones as well. It is obvious then that other methods for securing accuracy must be adopted. *The one inflexible rule of sampling is that the amount of ore broken into the sample must be proportional to the width of the orebody sampled.* For every inch of width, there must be included in the sample an equal quantity, measured by volume, not by weight, in order to represent the proportions that will later be actually mined. Where it is not feasible to secure the result with one sample, two or more fractional samples must be taken to represent one interval. It often occurs that the face can be trimmed at right angles to the dip from one wall part way to the other. From there on, however, the face is arched, and due to physical or economic reasons it cannot be squared preparatory to sampling. Under such conditions one sample is taken of the former portion and one of the latter. In each case however *the width sampled is measured on a line perpendicular to the dip* and not the length of ore face exposed, nor horizontally. Cases occur when a shell of ore is left on one wall of a drive and several feet of face must be sampled to represent a few inches of width. A few cases will be taken as types and in a subsequent chapter the method to be employed pointed out to serve as a guide.

Quantity Per Foot Sampled.—Under ordinary circumstances, most engineers in actual practice cut about 1½ lb. weight of ore per foot of width sampled. In many cases it is less, in a few cases it is more; in the latter instance, it is likely due to the softness of the material sampled. Engineers vary in their recommendations of the amount to be taken per foot of width, some even recommending as much as 4 lb. per foot. In his book T. A. Rickard[1] speaks of taking samples of 50 to 100 lb. weight. Except in the case of the softest kind of rock, this quantity is usually an economical impossibility. The amount of time required to cut such a weight is not only in most cases prohibitive, but the extra cost of preparing such large samples for assay, with the means usually available, not only adds unnecessary expense to an already expensive operation, but further weakens what is ordinarily the weakest link in the whole series of sampling operations, namely, the preparation of the mine sample for assay by successive reductions in size, dividing, and grinding the final pulp. Here, as throughout the sampling operations, the means at hand must be considered. At a property having complete mechanical crushing appliances, the size of the sample is of little consequence. In the most usual case, however, where only unskilled and ignorant labor is available for crushing by hand, and where time is an imperative taskmaster, small samples must of necessity be taken.

The actual sampling should be done by the engineer or his trusted assistants. In most cases an engineer works alone with one

[1] 'Sampling and Estimation of Ore.'

sampling assistant even on important examinations. Cutting samples is hard manual labor calling for patience and skill in using hammer and moil, and for this reason many engineers delegate the manual portion of the work to laborers. This is to be deprecated. The sampling forms the basis of the subsequent calculations and if incorrect the whole superstructure built on it as a foundation, is like a house built on sand. When intelligent white labor is available, it may be permissible to follow this procedure, but in most parts of Spanish America, or countries where only colored labor is available, the actual cutting of the sample should be done by the engineer or his assistants.

Engineers as a class are not accustomed to manual labor. Weeks and months of physical inactivity due to confinement in an office make a man's muscles soft and the first few days of physical effort trying ones. Whatever rules may be laid down, the tendency under such circumstances is to make the sample as small as consistent with accuracy. It is a remarkable thing how a man's viewpoint changes, when he is underground moiling infinitesimal fragments from an obstinate lump of hard quartz, that 99 times out of 100 has no value. Even half a pound of ore per foot of width sampled has often been made to go, and without vitiating the final result.

No arbitrary rule can be laid down. Sampling operations must be guided by the engineer's experience; each proposition must be studied separately, even each individual sample, and the work executed accordingly. It is well enough to state a specific weight per foot, but in practice actual results may vary greatly from the theoretical. An important factor is the character of the material to be sampled, hard ore is more difficult to cut than soft, besides it is easier to get a proportional cut from hard ore by means of a small sample, except in cases of blocky ground. In blocky ground it may be that only large chunks can be got conveniently, so that the weight of the sample is greatly increased. In soft ground, if it be brittle, samples of any size can be obtained; if it be clayey or sticky, or partly clayey the tendency at times is for the ground to come away in lumps, necessitating the taking of a similar quantity of the remaining ground, and in that way making a large sample. The character of the valuable mineral composing the ore largely governs the weight of sample. This, as well as other points, will be taken up under separate headings.

Interval Sampling.—It is customary in sampling a mine to take samples at regular intervals. We may express this operation by the term 'interval sampling,' although, curiously enough, I think it has never been given a name before. The object of sampling at regular intervals is two-fold. It permits of greater ease in calculating the average value of the ore and is a preventive against the human failing to sample the most likely looking and richest spots, if not restricted by an arbitrary rule of this character.

One result of the optical inspection or 'pick analysis' previously carried out is to enable a better determination of the frequency of the sampling interval. This varies according to the character of the material to be sampled. In high-grade gold ores or where gold occurs erratically in narrow veins, the interval between samples may be not more than two feet; on the other hand, in disseminated copper deposits, accurate results may be obtained from churn drill holes 200 ft. apart. In ordinary cases of medium or low-grade gold ores a sampling interval of 8 to 10 ft. is a usual one, with 10 to 12 ft. in ordinary copper, lead, or zinc deposits. No rule can be laid down, as the engineer must use his own judgment in each instance and be guided by his experience.

Having decided on a sampling interval, which for purposes of discussion we will assume as 10 ft., the engineer marks the place where the first sample is to be taken, in the manner already described, and definitely locates it, if possible with reference to some survey station, or otherwise to some permanent underground working, such as a cross-cut or winze. From the first sample, the intervals are marked with the whitewash every 10 ft., and at each one of these places the entire width of ore exposed is sampled.

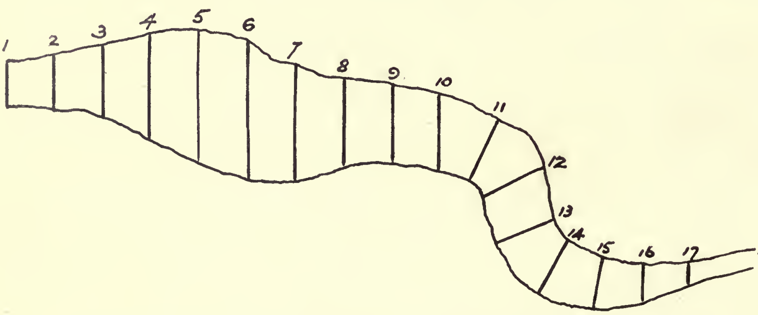

Fig. 8. METHOD OF SAMPLING A CROOKED WORKING.

Samples should be taken in a vertical plane at right angles to the dip. The dip is a line perpendicular to the horizontal line denoting the strike, and where local changes in the strike or dip are encountered, an adjustment must be made to meet the altered conditions.

Where development workings have been carried along one wall of an ore deposit, the almost invariable result is that such working is more or less tortuous owing to the natural deviation of the walls from a straight line.

Fig. 8 illustrates what is by no means an uncommon occurrence, a lode with frequent pinches and swells and a big bend. In the sketch these irregularities are somewhat crowded together for sake of illustration. The dip of the two walls is not always the same and the problem is, how shall the samples be taken. This is shown by the sketch. Samples 1 to 10 are in parallel vertical planes at right angles to the strike, although the line of sample is not in each particular case at right angles to the apparent dip, as indicated by the walls. From 11 onward the sampling interval is still the same ten feet, measured along the centre of the drive, but due to its curvature, the planes of the sample cuts are not parallel either to each other or the previous lot.

Where a mine is timbered fresh difficulties are encountered. In a case where a cap and posts are used without lagging, if the sampling interval comes opposite a set, the sample must be taken on one side or the other, and the succeeding samples continued at the regular interval previously marked out. In closely timbered mines, where lagging is used, denoting heavy ground, the lagging should be removed to permit the samples to be taken. If this is not possible owing to the danger of a fall of ground, nothing remains but to omit the sample. Cases of the latter sort are fairly rare, although sometimes the management may object to the cost of removing and replacing timber, or for purposes of deception, overrate the difficulties connected with such work. The engineer should in all cases satisfy himself on this point and if he believes there is no danger, he should insist on access to the faces to be sampled. Where one or more samples in succession must be omitted on account of timber, the question arises as to the assay value and width to be assigned to that portion of the working. Either it must be taken as without value or included in the average of the level as determined without it, perhaps allowing a certain deduction as a factor of safety. The plan to be followed must be determined by the engineer, taking into consideration the character of the deposit, its geology, the mode of ore occurrence, kind of mineral, grade of ore, etc., and the manner in which the mine is opened. In marking out the sampling intervals, not more than the day's work should be done, so that when sampling is resumed, precautions can be taken to prevent being salted.

Fractional Sampling.—When an orebody is more than four or five feet wide, or where there are two or more bands of ore of different grade or character, it becomes necessary to take more than one sample at the particular interval, and such operation is generally termed 'sectional sampling.'

This expression is objectionable to me, because the word 'section' as generally used in mining, implies the projection of a portion or the whole of the mine workings on to a vertical plane, whereas in the sense of sectional sampling, the word is borrowed from civil engineering, where it refers to a definite portion, such as a section of road. As opposed to this meaning, a sample of ore is taken in a mine, theoretically to represent a true cross-section of the orebody. This may be explained as follows: If a vertical plane be assumed to pass through the orebody at right angles to the strike, at the point where it is desired to secure a sample, then the intersection of this vertical plane and the mine working determines the line along which the sample is to be taken; thus, there is obtained a *section of the orebody* and to speak of a fractional part of such periphery as a section is technically incorrect.

Again, if more than one sample is to be taken from any particular interval, each one is merely a fractional part of the whole, because it is only of temporary interest and has been taken as a makeshift, to meet physical conditions beyond the engineer's control; besides which the individual results cannot be utilized, until they are combined and weighted proportionately to the width they represent, in order to arrive at the assay value of the whole interval. For this reason I consider that the logical name for this operation is 'fractional sampling,' and trust that the term may be adopted to clear up the misuse made of the expression now in vogue.

Fractional samples are taken in wide orebodies, and in cases where the ore occurs in bands of different grade or character, as, for instance, in a copper lode with a high-grade streak, of say 20 to 30% ore occurring adjacent to low-grade ore, of say 2 to 4% copper, or where a band of hard ore such as quartz, adjoins a soft ore such as decomposed irony or clayey material.

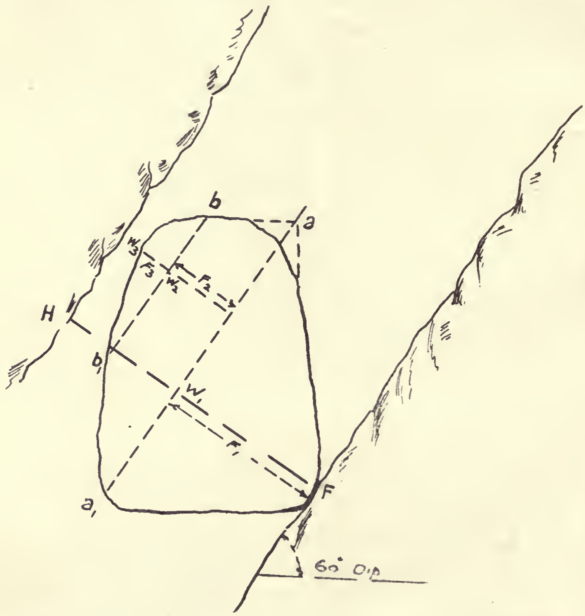

Fig. 9. INCLINED OREBODY, ONE WALL EXPOSED.

The most usual reason for taking fractional samples is because the irregularity of the mine working prevents, even in comparatively narrow veins, the cutting of an accurate sample in one operation. A variety of cases exists, but a few typical samples will suffice to illustrate the general methods to be employed to meet the varying conditions that may arise.

Case 1. Inclined orebody, only one wall exposed.

The illustration (Fig. 9) represents a drift following the footwall of an orebody, which is wider than the drift. The width of the ore is represented by the distance *F—H* perpendicular to the dip. As only one wall is exposed and that for perhaps no more than a few inches, and because nearly all orebodies pinch and swell, with the result that this face may be at a point of local variation from the general dip and strike, it is obviously impossible to accurately determine the dip from such a small surface.

The pick analysis, which has been previously carried out, should have enabled the engineer to determine the average dip of the lode and in a case such as this, the dip must be arbitrarily assumed and measurements based accordingly. This may lead to a certain amount of inaccuracy, but if the preliminary inspection has been intelligently done, the error, if any, will be negligible. In any event, it is the best solution possible under the circumstances, and such errors as occur may safely be counted on to balance one another.

It is obvious from the sketch that the ore cannot be sampled on a line at right angles to the dip, neither is it practical to sample all around the exposure in one cut with any likelihood of getting an accurate sample; the length of periphery exposed in different parts varies so greatly to the width represented, that it is not humanly possible to do so. Therefore, the exposure must be sampled in fractions. The upper right-hand corner must be squared up with a pick to the point *a* and three fractions marked out by the points *a* and *b*.

The first sample *F1* will be cut beginning from the footwall at *F* following the side of the drive to the point *a*, the second is taken in the back of the drive from the point marked *a* to *b* and the third from *b* to *W3*. It will be seen that sample *F1* is taken along a straight line on the footwall side of the drive and, therefore, proportionate amounts from this face will represent proportionate amounts of the width sampled. It will be observed, too, that this cut includes more than half the entire width of the ore and will affect the final result, or average value, in like manner. *F2* is to all intents and purposes a straight line as well, but represents a lesser width. *F3* represents only a small fraction of the entire width and the sample is taken along a slightly curved surface. Although it is desirable, it may not be possible to trim the corner as at *a* to take the sample along a straight line.

In most mines the drives do not conform to the ideal section shown in the sketch, Fig. 9, and this last sample, *F3*, is more often taken along a curved line than along a straight one. Care must be observed not to go beyond the point *W3*, where the drive is nearest the hanging wall, especially in a mine where there are high-grade streaks, as otherwise additional amounts of a band of ore already sampled will be obtained, giving an inaccurate result.

It is well to point out that under usual conditions the strata or bands, if any exist, will be more or less parallel to the dip of the orebody, and sampling across the exposed ends means the inclusion of almost the exact width, wherefore any irregularities of surface are not so likely to vitiate the accuracy as in the case of *F3*, where in the corner of the drive the face exposed may be nearly parallel to the hanging wall, and if care is not taken a large percentage of the fractional sample would come from an extremely small proportion of the width.

Case 2.—Highly inclined vein, one wall exposed, roof partly fallen.

When a drift is untimbered it may happen that a block of ore will fall out of the roof, leaving a gaping hole. This gives the sampler trouble. The indentation is perhaps a couple of feet deep, irregular and in hard ground.

A general rule cannot be given; the method to be adopted can only be indicated by a concrete example, as in the case illustrated (Fig. 10). An endeavor must be made to trim away the projecting points *X* and *Y* presenting three straight faces *F* to *a*, *a* to *b* and

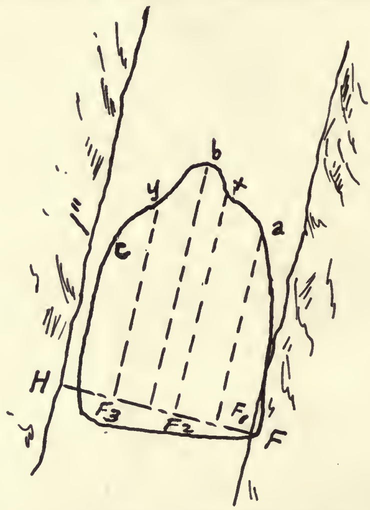

Fig. 10. HIGHLY INCLINED VEIN—FACE OF ROOF.

b to *c*, from which will come samples *F1-F2-F3* respectively and taken in the way already described. Where the character of the ground does not permit this operation it may be necessary to take five samples, namely *F* to *a*, *a* to *x*, *x* to *b*, *b* to *y* and *y* to *c*.

It can readily be seen that the character of the ground affects the method of meeting the problem. If the ground be firm, it can be picked down or even blasted. On the other hand, it may be too hard to be broken by pick or hammer and yet so loose that blasting is out of the question. It may be heavy ground unsafe to disturb even with a pick and requiring to be handled very gingerly. Under ordinary circumstances, however, the condition of the ground will permit of trimming the face in such a way that an accurate sample can be obtained by following one of the methods suggested.

Where the orebody is vertical *F1* will represent only a small portion of the entire width, in fact, only a shell of ore, and care must be exercised not to exaggerate its width in measuring, especially if there be a rich footwall streak. In this case, the length sampled and consequently the weight of the sample exceeds that of *F2* or*F3* despite the fact that the latter two, owing to their width, affect the average value to a greater extent. With an orebody having a flat dip *F1* becomes the most important fraction.

Case 3.—Irregular roof.

Fig. 11 represents an almost similar case to the preceding one. Here we have a highly inclined vein, where a fall of roof has taken place in such a manner that the point *a* is almost as close to the footwall as the lower right-hand corner of the drive, and although a length of perhaps six or seven feet may be sampled, the actual width of vein represented amounts to perhaps only a few inches. In a case of this sort it is very easy to make an error in measurement.

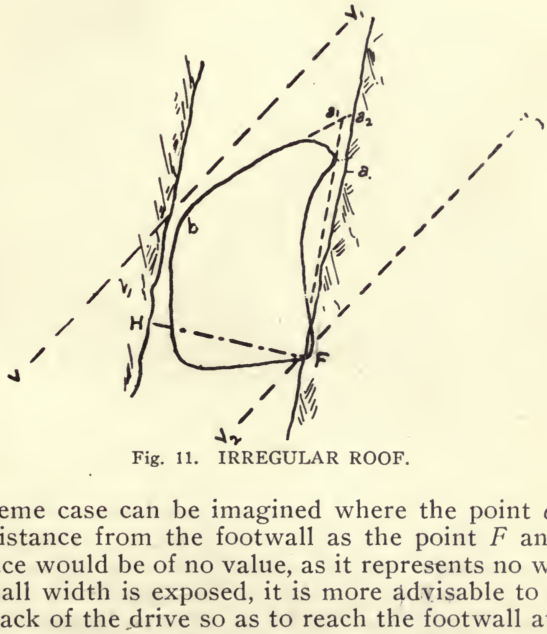

Fig. 11. IRREGULAR ROOF.

An extreme case can be imagined where the point *a* is exactly the same distance from the footwall as the point *F* and a sample from this face would be of no value, as it represents no width. Even where a small width is exposed, it is more advisable to dig out the ore in the back of the drive so as to reach the footwall at that point and thus permit the sample being taken in the back of the drive instead of along the side. The ground should be broken down along the dotted line to *a1* or to *a2*, if possible, and only one sample taken along the back of the drive from *a1* to *b* to represent the entire exposure. It will be seen that from *b* down to the floor represents a portion of the orebody already sampled.

A variation of this case is exhibited by the broken lines *V—V1*, *V2—V3* representing the same occurrence in an orebody dipping at a flat angle. Then the sample along the footwall side of the drive from *F* to *a1* would represent practically the entire width of the exposure and a sample from *a2* to *b* would be useless.

Case 4. Sampling at a Rise.

If the sampling interval comes opposite a rise it is usually impossible to secure a sample without a break or offset in the line of sampling—$F1$ being taken in the rise and $F2$ in the back of the drive. Often at a rise heavy timber is a necessity, making it impracticable to sample at the proper interval, then as a matter of convenience a sample is taken on either side of the rise at half the regular interval and proper allowance made in the subsequent calculations.

Where a sample is taken at a rise, sample $F1$ will extend along its side from the footwall at F to a and the width will be a—$a1$. Sample $F2$ would be taken across the back from a to b, the width sampled being b—$b1$. If the orebody were wider than the drive then $F2$ would extend from a to c after squaring up, or a third sample $F3$ would be taken did the conditions call for it.

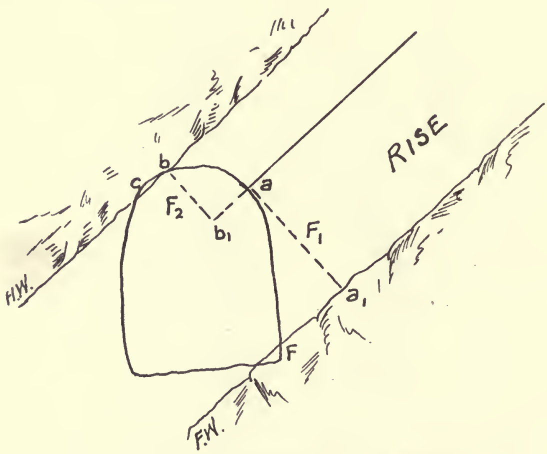

Fig. 12. SAMPLING AT A RISE.

The sampling of wide orebodies made up of bands of material of different grade and texture often presents a troublesome problem, and the sampler must use judgment to secure results that will enable a proper estimation of the mining possibilities. An inexperienced engineer, one who did not understand the practical side of mining, would be hopeless with a problem of this kind. Take, for instance, a deposit occurring in slate or in a much fractured granite and composed of bands of ore, varying in width from a few inches to as much as six feet, the total width being say 60 to 100 feet.

It may be accepted as a fact, that in a deposit of this character, the separate bands are not likely to be continuous over any great length along the strike, but that they will join and part company and will from time to time be separated by horses of country rock. The sampling must be done to afford evidence, whether it be more economical to mine the entire width as it stands, with subsequent hand picking, both underground and on surface, or whether only the bands, which of themselves have a minable width, can be mined alone, and the remainder left standing as pillars. It is necessary to fractional sample this body, the size of the individual fractions depending on the local conditions. As a result of such sampling, it may even be found that the more profitable result can be obtained by mining the orebody by a milling, shrinkage stope, or caving method without any sorting. A similar problem arises in the samples of underground development faces in disseminated copper deposits. The necessity for experience is evident.

Another case in point is the South African custom of sampling the rich leaders separately, including a measured portion of the country on either side. If two 'reefs' are close enough to be worked together, the intervening country rock is taken into the calculation of the average value as it dilutes the grade of the ore in direct proportion to its bulk. In sampling wide bodies of ore, experience in that great mining field has shown that fractional samples of two feet width give the best results. This width has not been selected on account of any irregularities of face but rather as a means of determining the distribution of gold in the ore and probably to counterbalance the unavoidable inaccuracies in the routine sample of operating mines.

These few cases by no means exhaust the various conditions that may present themselves, but are deemed sufficient to draw the attention of younger engineers to the fact that sampling cannot be done in a haphazard manner, nor according to hard and fast rules, but that the method adopted must suit the conditions encountered.

Sampling Rises and Winzes.—It is sometimes exceedingly difficult to sample rises and winzes especially in vertical or highly inclined veins. Probably in the majority of cases the ladders have been removed, there is little or no timbering, and there is no windlass handy. The collar of the winze may even be in such bad condition that a windlass can be put in place only with exceeding difficulty. The difficulties of inspection or sampling should not deter the engineer from doing his work well. In many instances these are increased by the owners' representatives from no charitable motive. Unless an engineer is willing to take uninspected winzes and rises into his calculations as worthless, they must in all cases be examined and sampled.

Before undertaking to sample a rise or winze, it must be carefully subjected to pick analysis. The work must not be shirked because of the difficulty of moving about and inspecting the orebody in a working of this kind, when suspended in a bucket, boatswain's chair, or when balancing oneself on shaky planks or poles laid across uneven timber. The optical inspection is essential. It enables the engineer to determine such conditions as, whether the full width of the orebody is exposed, whether the working follows one of the walls, or is tortuous, and affords evidence as to the exposure of any known high-grade streak, as well as any change due to geological causes, such as the occurrence of the values in floors.

Rises and winzes are usually made for one of three reasons, namely: (1) Exploration; (2) ventilation; (3) blocking out the orebody to facilitate subsequent mining operations. If the purpose is for exploration, then in all probability the rise or winze will follow the orebody in all its twists and turns, and the tendency will be, unless the vein is very wide, to expose its full width.

When either the second or third object is to be attained, it is likely that the working will be more or less straight, in which case the orebody may be left partly or altogether in one of the walls. The sampling must be done to meet whichever one of these conditions is encountered, and the interpretation of the results is effected in a like manner. When the orebody is wider than the rise or winze and only a portion of its total width is exposed, unless carried along one of the walls, the working may cross and even recross a given run of valuable ore. In a case of this kind sections of such workings should be carefully platted from actual surveys and the sampling results carefully interpreted. On the other hand, if the whole width of the orebody is exposed, samples taken at regular intervals will furnish results that may be used for calculating the general averages required for arriving at the estimates of tonnage.

When only a portion of the orebody is exposed, the sampling results cannot be used for this purpose, but merely to serve as a guide and as some evidence of the persistence of values from one level to another, and this must be kept in view while the sampling is being done.

When only one wall is exposed, the utmost caution must be observed in case a high-grade streak is included in the portion of the orebody that can be sampled. When the ore passes in and out of the winze or rise, because either the vein or the working is tortuous, such ore as is exposed should be sampled and the width noted, and the results used as a guide in sizing up the deposit. When the wall rock is sampled, it must not be included in the same sample as the ore, but must be given a separate number and assayed separately.

In some orebodies there is a tendency for the valuable ore to occur in floors or flat-pitching shoots, and the vertical development openings afford information of the most useful character. In limestone deposits the occurrence of ore along floors or flat-lying beds is quite common and the evidence afforded by winzes or rises in this type of deposit is of the utmost importance. In cases of this kind sufficient information may even be obtained by optical inspection to form an adverse opinion and thus obviate the expense of sampling the mine.

Particular attention should be paid to winzes below the bottom level. When developments in the bottom are bad, it is a common trick to allow winzes to fill with water, or they may even be intentionally filled with waste to prevent inspection. The engineer cannot be too cautious on such an occasion. A mine with a bad bottom is usually a bad bargain. Once eliminate the possibilities of extension in depth, then the mine has no prospective value, and if you eliminate the prospective value of a mine, the chances of big profit and long life are gone—factors which play an important part in mining operations and which cannot be overlooked.

Sampling Cross-cuts.—Occasionally a mine is opened up by cross-cuts only, the drifts being carried in the country because greater speed can be made. The reliance that can be placed on the results from development openings of this kind depends on their frequency, at the same time taking into consideration the character of the ore deposit. Under no circumstances is the information afforded as satisfactory or reliable as is the case where the drifts have been carried in the orebody. Particularly is this true with gold ores, but even with base metals where there is any tendency for a segregation of the mineral, care must be exercised in interpreting results. The character of the mineralization, therefore, to a large extent governs the manner of sampling, not only where an orebody is opened only by cross-cuts, but also in the sampling of any cross-cuts whatsoever.

For purposes of discussion, assume a case where the full width of the ore is exposed. The ore may be sampled along the sides, back or floor. Sampling the floor is usually out of the question on account of its greater inconvenience. The most common plan, especially if the drift is in the orebody, is to sample across the back, because the full width can be sampled without a break in the cut. When sampling the sides, a break occurs at the junction with the drift, and the sampling must be finished across the back of the drift. The method to be followed is governed by the sampler's experience.

In sampling deposits of the massive type, such as disseminated copper deposits, or bedded lead-zinc deposits, the best results are obtained by dividing the cross-cut into 5-foot intervals and making a continuous cut along the back and two sides and combining them for one sample. If there is a good bunch of ore in one place, it is easy enough to get a false result by taking more than its due proportion.

One must be careful to get as nearly as possible the same weight of ore from each of the three cuts. In fact, the precautions already set forth at length elsewhere are to be observed, and if advisable each fraction may be assayed separately and the results averaged in the usual way.

I have seen a cross-cut sampled in a sort of spiral, namely, from the floor on one side, up the side, across the back and down on the other side, the cut advancing and finishing 5 ft. or 10 ft. in advance of where it started. The only thing that can be said of that method is, it is different from the generally accepted method, it is hard to see any increased accuracy resulting therefrom, while on the other hand, it is more difficult to cut.

In inclined orebodies, and they include the vast majority with which engineers have anything to do, sampling across the back or sides in a continuous groove involves taking horizontal measurements, unless by means of a string the angle of inclination is laid off and the true width of each fraction measured as it is taken. If the horizontal measurement be taken, the true width should be calculated and set down in the note book for use in the subsequent calculations and assay plan. This is determined by the following formula:

True width=Horizontal measurement x angle of dip.

Another way is to figure out beforehand the horizontal distance corresponding to the true width of the fraction it is desired to take and to lay off the calculated length in the cross-cut before sampling,

thus let h = the horizontal width,
w = the true width of the fraction,
a = angle of dip.

$$\text{then } h = \frac{w}{\text{sine } a}$$

In orebodies where there are strata or seams, more or less parallel to the dip of the orebody, fractional samples are sometimes taken across the true width, the successive fractions making a staggered line, like a flight of stairs. This is not to be recommended. Wherever possible samples should be cut from a continuous groove. Seams may contain ore of high value and by staggering the samples there is greater danger of getting an excessive amount of the high grade.

The calculation of tonnage in an orebody opened by cross-cuts alone is a proceeding that often is attended with considerable risk. Even where there is no attempt at deception on the part of the owners, cross-cuts are often put in at likely looking places and the resultant calculations are then in excess of the breaking value of the ore. Horses of rock and bunches of low-grade ore have a habit of making in most unexpected ways.

When the orebody must be studied from cross-cuts alone a certain amount of misconception may arise on account of parting planes and seams dipping in misleading directions, so that the samples taken are not gauged according to the real dip of the orebody. Especially is this true where the dip of an orebody changes or reverses. A single cross-cut in what appears to be a wide orebody may be on a branch vein and a few feet on either side of the so-called cross-cut the valuable ore may terminate. When an orebody has been drifted on, cross-cuts give valuable information; besides, corroboratory evidence can be obtained by drill holes from the sides of the drifts. In the massive type of deposit, care must be exercised in calculating tonnages and values near the edges of the deposit. It is particularly in cases of this sort that blocks of barren ground may be included.

The principle involved is that cross-cuts alone do not prove the continuity of the orebody in the same manner that drifts do and therefore cannot be relied on to the same extent. The distance between cross-cuts is an important factor and the greater their frequency the more reliance can be placed on the results obtained. In any case caution is necessary.

Measurement of Width Sampled.—It has been stated that the widths sampled should be measured at right angles to the dip of the lode. Referring to Fig. 9, it will be seen that there is only one fixed reference point, namely the footwall where it is exposed at *F*. Sample *F1* was cut along the side from *F* to *a* and the width actually sampled is represented by the distance *F—W1*.

The only accurate method of measuring that width, is to stretch a string from *a* parallel to the dip and measure the distance between *F* and *W1* by means of a steel tape. The end of the string at *a* may be held by the sampler's assistant or secured with a nail—the lower end against the opposite side of the drive may be held in place by the hand or by winding the string around a hammer, pick, or other tool.

The string *a-a1* can be accurately lined up by going off a few feet and gauging the angle of the string, which is illuminated by a candle held behind it. The width of *F2* can usually be determined by using another string or by lining up a piece of steel. Finally the width of *F3* is taken by stretching the tape to the farthermost corner and subtracting the width of *F2*. By this method there is less liability of error than if the measurement were attempted direct. This method is accurate beyond question and immediately gives an absolute result, regardless of any twists, turns or rolls in the formation and there is no overlapping of measurements.

Another way that is used by some engineers is to measure the widths by means of sticks. This is not only cumbersome and apt to give inaccurate results in a narrow mine working, perhaps necessitating its rejection, but the proper gauging of the width is difficult, and therefore there is more liability to error. In mine sampling, every safeguard it is possible to employ to avoid mistakes should be used; as there are enough uncertain factors in the work that cannot be provided against and the engineer should not multiply them.

Some engineers take the horizontal measurement. In wide veins or in a case such as Fig. 9 (p. 34), it is not only impracticable but likewise apt to be inaccurate, although in veins narrower than the drive it may be applied. The theory underlying horizontal measurements is that all the measurements are proportional in the same degree to the true width, but the method gives incorrect results, where the dip of the orebody changes along its strike or dip, as the following illustration will demonstrate.

In calculating the volume of a block of ground for tonnage purposes where the true width of the ore has been measured, the area is obtained by taking half the sum of the two widths on either side of the block, which is multiplied by the distance between them as measured along the plane of the orebody.

Where horizontal measurements are used a cross-section shows a rhomboid instead of a rectangle and to get the area half the sum of the two horizontal measurements is multiplied by the perpendicular distance between. Where the orebody rolls, the volume thus obtained is incorrect, because this method is based on the theory of proportional length between vertical and hypotenuse. When the orebody takes a roll, the length of the hypotenuse may be considerably augmented.

Horizontal measurements are no doubt easier to take, but they do not give results as accurate as those obtained by measuring the true width direct. It may be claimed for horizontal measurements that the results are on the safe side, nevertheless, the engineer must guard against inaccuracies of any kind, for, as already emphasized, if it is desired to use a factor of safety, it must not be left to chance.

CHAPTER V.

PRECAUTIONS NECESSARY IN SAMPLING.

The foregoing paragraphs have dealt with the principles underlying sampling operations, and a few of the problems that may be encountered in mining operations have been illustrated. This chapter will deal largely with certain mechanical operations connected with the handling of the sample underground. It has been pointed out that the place where the sample is to be taken should be marked by whitewash or chalk; that the cut should be a straight one; should be representative of the width sampled, and should be caught in a duck bucket or other similar receptacle.

Sampling a homogeneous material or one where the valuable ore is evenly distributed through the gangue, does not require the same amount of care as in the case where it is spotted. For example, in sampling an iron ore deposit or low-grade copper ore, an accurate sample can be secured in most instances by smashing down the ore with a mining pick into a large canvas sheet, although such a procedure is not recommended.

Low-grade copper and iron ore deposits are bought and sold on the results of drill-hole samples, and from a sampling standpoint are in a class by themselves. Under ordinary circumstances the difficulties of sampling that present themselves to an engineer are largely due to the brittleness of the valuable mineral in a harder gangue, and also to the occurrence of erratic values in the ore. For this reason the use of the sampling bucket gives more reliable results than the canvas, and largely represents the difference between sampling for valuation per se by an independent engineer, and the methods employed in taking the routine samples by the mine staff in going mines.

Where the ore contains sulphides, the blow of the hammer on the moil jars the ground, and has a tendency to shake down the brittle mineral. The larger the area exposed to receive the falling particles of mineral, the greater the likelihood of contaminating the sample, with what is usually the richest constituent of the orebody. Many people state that a mine cannot be sampled to show its stoping value. If sampling is properly done, there is no reason why it should show a higher value than can be mined, barring the admixture of the wall rock with the ore, for which allowance should be made.

In operating mines, experience shows that the daily mine samples must be reduced in value by 10 to 15% to show the mill head value or assay value at the face. This, in my opinion, is due to the necessity of hurry in taking daily mine samples, for which reason less care is possible. Such samples come from the development faces and stopes, and while the sampler is at work, the miners are compelled to be idle, and because assay results are required the same day, speed is an essential and this error due to the method employed is allowed for by a factor of correction. While the factor employed may be incorrect in individual instances, these errors are presumed to balance one another.

The usual method of taking daily samples, is to lay a large canvas on the ground on which the sampler steps, and by means of able-bodied blows with a pick, a portion of the ore in the face is broken down into the canvas. Inevitably by this method chunks often fall from the face into the canvas, become mixed with what is already there, and make an inaccurate sample. Above all, however, the chief error originates from the brittle and valuable sulphides being jarred down into the canvas from a considerable area of the face by the force of the blow.

In mine examination work, the sample is carefully cut by moil and hammer, and as much time as necessary to secure an accurate result is taken by the engineer. In many places, not over ten or twelve samples per day can be gotten by a good sampler with one or two helpers. In the Broken Hill South Blocks mine, the ore was so hard that a sampler with two skillful miners working double-handed was able to secure only five or six samples a day. This is, of course, rare. The result is that in that district daily face samples are not regularly taken in the mines.

In cutting a sample in mine valuation work, if a piece of ore flies beyond the bucket, it should not be picked up from the floor, but another piece chipped from the face. Picking pieces up from the floor is likely to lead to error and danger of salting. If too large a quantity accidentally comes away, and falls into the bucket, nine times out of ten the sample should be thrown away and a new start made. This may involve considerable additional work, but should not be neglected except in the sole case where the ore has come away in a single chunk, and can be picked out, but even this is bad practice.

Care should be taken not to include too much of the softer seams, if there are any. These, too, may represent the richer portion of the ore. The greatest caution must be used in sampling high-grade streaks, and if possible they should be sampled separately. Erratic high assays often represent carelessness in sampling. Hard ground being more difficult to cut, offers a temptation to the sampler to take a smaller quantity than required to represent it, and great self-restraint is oftentimes necessary on the sampler's part to keep him from moving ahead before he has secured a proper quantity from hard ground, urged on as he may be by tired muscles.

A quantity of the wall rock should be included in every sample, not only to make sure that the limit of valuable ore has been reached, but also because when stoping the ore is bound to become diluted to a certain extent by falls of the country into the ore.

In orebodies where the ore occurs in strata or streaks of different texture and hardness, a condition presents itself that often makes accurate sampling difficult. In the preliminary work preceding the active sampling operations, if an assay plant or other means of testing is present, a few samples may perhaps be taken to advantage for the purpose of determining the distribution of the valuable minerals. It may be found that a certain kind of quartz carries most of the gold, in which case it emphasizes the desirability of sampling such quartz separately.

In the 'L' mine the vein is composed of hard quartz, in places honeycombed and like gossan, due to the oxidation of pyrite, a yellowish clay and manganese oxide, which occur in a more or less banded structure. Careful tests failed to point out any one of these as carrying an abnormal amount of precious metals as compared to the others. Had there been a compact band of quartz followed by a band of soft ore, the work would have resolved itself into taking a sample from each. As a matter of fact, however, no regularity exists. Oftentimes a narrow seam of quartz occurs embedded as it were in the clay, which, when struck, would retreat into the yielding material. So marked was this that in instances it was necessary to complete the remainder of the cut and then dig out the quartz and break off the desired amount on the floor of the working. In a case of this sort, however desirable it may be, it is manifestly out of the question to sample this small seam of hard material separately from the rest. It did not assay very differently from the rest, and while an error may creep in, it is reduced to such a small percentage that it does not appreciably vitiate the result.

In this mine great liberties were taken in the sampling by various engineers who had previously examined it, yet the average values obtained did not materially differ one from the other. While this is pointed out, it is by no means intended that lax methods may be employed. The exception but proves the rule. In sampling this vein in places the quartz was so hard that it would take an hour to moil 12 to 15 in., whereas the clayey and manganiferous material came away so easily that only the lightest blows from a prospecting pick were permissible to prevent knocking down excessive quantities. It is in material of this character that a prospecting pick may be used in preference to a moil and hammer. This, too, is one of the cases where an extreme condition exists, and where in most cases caution is required to prevent inaccurate sampling by taking too large a proportion of one or the other kind of ore. At times the soft material would come away in a chunk; at others too large a piece of the hard ore.

In orebodies of the type of the Broken Hill deposits, the valuable minerals, i. e., the comparatively soft and brittle lead and zinc sulphides (galena and blende) occur in a gangue of garnet rock. Garnet rock is one of the hardest rocks that the sampler ever has to encounter and under the moil is most refractory, even when using a double-hand hammer. Here we have two extremes, and unless great care is used, the samples will all run high on account of the excessive proportion of the valuable friable mineral, which is not only easier to break down, but shakes into the sample from above at every blow of the hammer.

Bodies of solid pyritic material would seem to offer the simplest sort of problem for sampling, yet even in this class of deposit care must be exercised against unconscious salting, where the iron pyrite contains copper pyrite. As usual in this class of deposit, amount of copper is not evenly distributed throughout the mass, but it occurs segregated in bunches, streaks, nodules and lenses. As is well known, one of the simplest means of differentiating between iron and copper pyrite by optical inspection is the test for hardness, ordinary pyrite being unscratchable with the knife blade, whereas copper pyrite is easily scratched. Hence in sampling in this class of deposit, care must be exercised, because on account of the softness of the copper pyrite, excessive amounts may be taken, thereby unduly raising the copper contents of the sample.

Another case which might be pointed out is the occurrence of low-grade silicious streaks or horses in the midst of orebodies of this class. These silicious horses often contain good percentages of copper and therefore constitute part of the orebody. And even in the case where from an optical inspection, it would appear that such silicious material does not contain sufficient copper to be classed as ore, it should in most cases be sampled, because when correlating the results of the sampling operations and deciding on a method of mining, it may be found more economical or even essential to stope these silicious horses with the remainder of the orebody. However, each case must be judged on its merits.

In pyrite masses there is usually a tendency for a segregation of the copper, with the result that often there is a rich and usually persistent streak along one of the walls. Sampling, therefore, must be carried on with a view of determining the value of this higher grade streak separately from the larger bulk of lower grade material adjoining it, because it may be found that when the final calculations are made the economic conditions may only permit mining this high-grade streak.

In sampling ore that is narrower than stoping width only the ore should be sampled. It is bad practice to include in the sample barren country rock to the full stoping width. Not only is it useless labor, but it hides the real width and value of the orebody at that point and possibly prevents a proper interpretation of results. In making the calculation for tonnages the narrow widths can be calculated to stoping width by reducing the assay in the proper proportion.

Where a streak of high-grade ore is opened in a working in which the full width of the orebody is not exposed, the assay of the sample at this point is too high, and therefore the assay value of the interval should be reduced. That ore of high value may occur erratically is well known, and in gold mines, where it does so occur, it is customary to reduce the high assays to the general average of their neighbors. In wolfram or mixed wolfram and tin deposits, the mineral is apt to occur in rich pockets or bunches surrounded by low-grade or even barren gangue. In such cases the erratic results must be averaged with the others to arrive at the stoping value of the ore. In some gold deposits high-grade streaks occur in conjunction with low-grade ore, the high grade containing visible free gold. It is perhaps needless to state that each band must be sampled separately and a great deal of judgment used in bringing the high-grade results into the calculations.

In the replacement type of gold deposits, where the precious metal occurs along cracks or fractures in the rocks, caution must be observed that a sample does not follow one of these lines of enrichment. The mineral may occur in parallel lines of fracturing within the orebody itself, which fact should be discovered by pick analysis, and the sampling done to meet that condition. In some deposits, especially in limestone, the ore is deposited in floors, or flat 'makes,' of which there may be several separated by unprofitable material. If this be discovered in time, the whole expense of sampling may possibly be avoided, although some of the richest mines known have deposits replacing soluble limestone beds.

If an engineer is conscientious he will endeavor to secure as much information as possible regarding the assay value and quantity of ore available. It is bad practice not to get information as to the character of the ore left in the sides of drifts, etc., where the orebody is wider than the workings. Where only an occasional bulge occurs it may not be worth while, but under ordinary circumstances not only should the exposed faces be sampled but horizontal drill holes put in the sides by steel and hammer and the chippings assayed. It is not only desirable on the score of the increased tonnage available, but also because in mining, this portion of the orebody will be broken down with the remainder and, if low grade, may turn a profit into loss. In case of wide veins where there are insufficient cross-cuts, diamond drills can often be employed to advantage, though not often available.

CHAPTER VI.

GEOLOGICAL FACTORS IN SAMPLING AND VALUATION.

In this chapter it is proposed to call attention to some of the geological factors appertaining to mine sampling and valuation not discussed elsewhere. That the geology of an ore deposit has an important bearing on the sampling operations is evidenced by the necessity for pick analysis, which has already been elucidated. Besides this it has a serious bearing on the valuation of the ore reserves, because a knowledge of the geology of the deposit is essential in forming a definite opinion. Its importance can therefore be appreciated. A mistake often made by engineers in mine examination work is that valuable time is lost making a more or less detailed study of the general geology of the district, that could be used to better advantage underground.

One thing should be remembered, namely, that the mining engineer is rarely an expert geologist and the geologist even more rarely a proficient mining engineer. The character of the work of each is different, as is the object to be attained. The work of the geologist deals not at all with the commercial aspect, but with the results of cause and effect; he traces out the geological phenomena and what may be expected therefrom. The mining engineer, on the other hand, is first and last concerned with the question of profitable operation.

In mine examination for valuation, only such geological features need be investigated as have a direct influence on the ore deposit, and under ordinary circumstances the general geology of the district has no direct influence on the value of the mine. Occasion may arise for the engineer to go some distance afield, in order to secure evidence bearing on the genesis of the deposit, or confirm observations made underground, but in general it may be stated that only the immediate geological aspects of the deposit need be considered. If we had an orebody completely outlined and delimited by development openings, its geology would be of no consequence to the mine valuer. It is only when questions of doubt arise as to the continuity of ore that the geological viewpoint must be seriously considered.

Aside from the association of precious metals with certain minerals in the orebody, the value of the ore may be affected by the nature of the surrounding rock, by faults and dislocations, by fracturing and fissuring, and by other well recognized phenomena. The influences of secondary enrichment often have a marked bearing. The character of the formation in which the deposit occurs is of great importance and is perhaps the first geological factor to receive attention. What kind of rock surrounds the orebody? At the surface, rocks are more or less weathered and if the weathering is at all pronounced, it becomes difficult to classify a rock from optical inspection. I have seen expert geologists at a loss to determine a rock under such circumstances.

The mining engineer is generally less expert in this work, yet in an attempt to show profound knowledge, or in order to attain an accuracy that is entirely unnecessary, he often over-reaches himself and shows ignorance. To prove this, all one has to do is to take two or three reports on one property by different engineers, and it will usually be found that if there is any chance to call the rock formation anything else than it really is, it will be done. At any rate, each engineer is likely to call the rock by a different name, although the financial side of the business may find all of them in accord. The character of the rock formation and the kind of rock is not by any means a guide to the persistence or richness of an ore deposit; and after all the important question is, can it be profitably mined?

I do not mean to deprecate the usefulness of careful geological work to the mine valuer, but merely wish to point out that a detailed geological investigation and accurate classification of the rocks may be unnecessary and serve no useful purpose in the work of determining the worth of the mine. If it can be done, so much the better. On the other hand, when an engineer is in doubt, his purpose will ordinarily be sufficiently served by the use of the following eight general groups:

Class.	*Kind.*
Igneous	*1.* Granitic Rocks. *2.* Porphyritic Rocks
Volcanic	*3.* Lavas—Light and dark
Sedimentary	*4.* Sandstones *5.* Limestones
Dark Colored Basic Eruptives	*6.* Trap Rocks
Metamorphic	*7.* Schists (including Gneiss) *8.* Shales and Slates

It is only in rare cases that the geological age is of the slightest interest in mine examination work. We know that in certain districts, gold deposits ordinarily occur in rocks of a particular age, but this is no proof that similar deposits in other districts will yield a profit, nor even that all deposits in the same field are of value. As the Cornishman says: "Where it is, there it is."

One of the most common statements made by people who have mines for sale is that the deposit is geologically the same in all respects as some famous producer. In the United States for years attempts have been made to interest capital in some occurrences of native copper in trap rocks in the Shenandoah Valley of Virginia. It is claimed for them that they are geologically the counterpart of the famous Calumet and Hecla lode. And they are, too, so far as the rock goes, but unfortunately the copper does not occur in commercial quantities.

During the South African boom it was claimed that hundreds of miles away other deposits similar to the Rand banket had been discovered. The West African banket forms a more recent spectacular illustration of the use that is made of similarity of formation. The difference in the value of the two deposits is well known.

Another assertion commonly made is that the property being offered is on the same strike as some other lode miles away, or that the ore opened up is a continuation of the famous mine on the other side of the mountain. Such a statement may or may not be true. If true, it does not necessarily increase the value of the ground under consideration. All orebodies have a definite length and speaking generally, beyond the profitable zone, there is little likelihood of finding another pay shoot, so that the prospect must not be overvalued on account of its proximity to a highly developed and prosperous mine in the neighborhood. Nevertheless, one of the greatest aids to the examining engineer is that afforded by studying other mines in the same district.

It may be taken as an axiom that in any given mining district, the same kind of deposit occurring under the same geological conditions, has been formed contemporaneously with its neighbors and geologically may be expected to yield similar results with similar development, although this does not necessarily mean the same financial results. If we refer to the history of Butte, Montana, we find that the gossan outcrops were formerly mined for their silver content and the presence of copper in the deposit was unsuspected for some years. On further development these outcrops proved to be leached copper gossan and in depth the copper which had migrated from above was found re-deposited in a highly concentrated form, the ores consisting of chalcocite, covellite, bornite, and other minerals rich in copper. The recurrence of the same phenomena in similar deposits in that district is a natural and justifiable deduction, but is not sufficient reason for assuming that a silver-bearing gossan 200 miles away, would also turn into high-grade copper ore in depth. The gossan may easily have resulted from the weathering of a body of iron pyrite, with which no copper was associated.

The importance of the geological evidence must be decided on its merits, and the keen observer may be able to see further into the ground than another less observant individual. It is only partly true that no man can see further than the point of his pick. Some men see and properly estimate indications unseen by others. I know of a mine worked to apparent exhaustion by one manager, who recommended his directors to abandon the property, whereas his successor, after studying the formation, drove a cross-cut less than 20 ft. and the property has been paying dividends for the eight or ten years that have since elapsed. In mine valuation it is not often that we dare capitalize indications of that kind, but in a case where the purchase is warranted by the ore reserves, a line of development may be recommended to the eventual benefit of the engineer's clients.

Many geological problems arise to perplex the mine valuer. In large mines, where the character of the orebody may be carefully studied in many hundreds or thousands of feet of development openings, the problem is simplified. The evidence is not always complete, yet the constructive type of engineer likes to form a definite opinion as to the future possibilities of the mine. In other words, he wants to feel that the mining risk assumed by his clients will be justified by the subsequent developments. No man cares to undertake a business that will merely return his original investment. The greatest skill is called forth in remote districts where the engineer's sole guide is his experience and there are no neighboring mines to furnish corroboratory evidence. Under such circumstances I have known more than one engineer to condemn a mine sooner than risk his reputation. Such practice is more than reprehensible.

With a mine having perhaps a large amount of development work above water level and the purchase price warranted by the ore reserves, a close study of the geology is, nevertheless, essential. The engineer is called upon to determine not only the likelihood of the persistence of the ore below the water level, but also whether its mineralogical character will so change as to render profitable treatment impossible. In the oxidized zone of gold deposits the presence of arsenic and antimony is often unsuspected as is zinc in carbonate of lead ores. Aside from the question of secondary enrichment discussed later, the liability of certain metals to concentrate at particular horizons must be ever kept in view.

On study it may be shown that the valuable ore is genetically connected with the intrusion of dikes, in fact, the dikes may have been the cause of the fissuring. Cross-dikes may cut off valuable ore altogether or may mark the channel along which the enriching solutions or gases penetrated, so that profitable ore is only found at the intersection of the vein and dike; then, too, they may cause faulting of the lode and sometimes an impoverishment of the vein. The selective action of ore-bearing solutions is well known, and this is most marked in limestone formations, where perhaps a seam may have acted as a conduit for the solutions which have spread out in one of the more soluble beds to form an orebody lying conformably to the stratification. This same phenomenon may be repeated at other points.

It is also well known that the same lode in passing from one kind of rock to another may vary in the character of its mineral content. The change from copper to tin in Cornwall is well known. Another illustration is the lode formation at Silver Islet in Lake Superior, which contained a wonderful deposit of native silver at that point, but on the mainland the same vein in different country rock contains no silver. The Camp Bird mine, Colorado, is another case in point, where development below a certain horizon proves fruitless. The valuing engineer must study the formation to learn of the likelihood of the lode passing into rock of a different character and the influence that such change may have on its value. In the monograph on Goldfield, Nevada, prepared by eminent geologists, and issued by the U. S. Geological Survey, the statement is made that in passing from one rock to another the value was likely to decrease. Subsequent work has proved this theory to be incorrect, and it is quoted as an illustration that unless the evidence is positive, it is unwise to draw conclusions of this kind. The main proof after all is the development work and it would seem that usually a statement to the effect that valuable ore will not continue beyond a certain horizon is apt to be falsified in cases where such a statement is based on theory instead of the evidence of actual underground workings.

The geology of contact-metamorphic deposits is particularly puzzling and the literature on the subject shows that duplicate geological conditions do not imply the same commercial value. Much has been written on the geology of ore deposits and justice cannot be done in the few pages devoted to it here. In solving the problems that present themselves, experience and judgment are necessary.

There are a number of useful works on Economic Geology that may be perused with benefit by the mine valuer. A knowledge of ore deposits is an essential qualification to him; and although the ability to accurately classify rocks in the field is not of the first importance, it is, of course, of some satisfaction to the engineer to be able to do so. The table on p. 55 is republished from the *Mining and Scientific Press,* vol. 99, p. 599.

There is a maxim in mining that practically all ore deposits, except those of iron, are subject to the influences of secondary enrichment, and that in the zone of secondary enrichment ore of higher value will be obtained than below it.

The effects of secondary enrichment on copper deposits are of course marked, and have been so active as generally to have removed practically all the copper from the portion of the orebody lying near the surface, leaving a well marked gossan. Below this mass of oxidized material is a zone of greater or less extent, containing a concentration of the valuable metals, often 20 or 30 times as rich as the original unaltered material, which when penetrated may be found to be of no commercial value. The low-grade disseminated copper deposits are startling illustrations of this fact.

Again the history of gold mining demonstrates that secondary enrichment is an important factor in raising the assay value of the ore lying near the surface to an important extent, for we find that all gold mines eventually pass out of the higher grade of ore, which has been the cause of early success, into ore of considerably lower grade. In the case of gold mines, however, the change, though well marked, is not so great as is ordinarily the case with copper mines, and provided the mine has been properly equipped with efficient plant, profitable operations may usually be continued for a considerable further period.

The object of pointing out these facts is to draw the attention of the engineer to the necessity of keeping them in mind during sampling operations, and he should endeavor to secure information bearing on this point. If a gradual enrichment is found from the surface to the

IGNEOUS ROCKS.

PLUTONIC.	INTERMEDIATE.	ERUPTIVE.
	GRANITE-RHYOLITE SERIES.	
Granites. Holocrystalline. Dominant minerals: quartz and alkali feldspar (orthoclase or microcline). Subordinate minerals: muscovite, biotite, hornblende, etc. *Aplite*, a granite with quartz and feldspar only.	*Quartz Porphyries.* Intermediate between the granites and rhyolites.	*Rhyolites.* Eruptive equivalents of the granites with same chemical composition. Quartz and feldspar, predominating minerals. Commonly contain more or less undifferentiated glass. *Obsidian*, is the wholly vitreous variety of rhyolite.

The difference between granites and rhyolites are structural and genetic; chemically and magmatically they are the same.

SYENITE-TRACHYTE SERIES.

The Syenite-Trachyte series differs from the Granite-Rhyolite series in being free, or nearly so, from quartz. All of these rocks contain principally alkali feldspars, with subordinate femic minerals, and often alferric species, such as hornblende, mica, etc.

PLUTONIC.	INTERMEDIATE.	ERUPTIVE.
Syenites. Resemble the granites in their deep-seated plutonic origin and in being holocrystalline.	*Syenite Porphyries.* Intermediate forms between the syenites and trachytes, analogous to the quartz porphyries.	*Trachytes.* Like the rhyolites, they are eruptive rocks.

NEPHELITE SERIES.

These are transition rocks from the syenites and trachytes proper, to the phonolites and nepheline syenites. They occur in plutonic, intermediate, and eruptive groups like those above. Quartz is absent, and lenads, or feldspathoids (feldspars deficient in silica) replace the feldspars to a greater or less extent.

Phonolite is commonly made up of orthoclase, nephelite, and pyroxene.

MONZONITE GROUP.

The Granite-Rhyolite series of rocks, and the Syenite-Trachyte series also, are defined by the predominance in them of alkali feldspars, and commonly of orthoclase.

The Andesite-Diorite series (see below) is characterized by plagioclase feldspars.

Between these series are all sorts of gradations known as *monzonites*. All these rocks carry orthoclase or anorthoclase and plagioclase in approximately equal amounts, with or without quartz, and with smaller amounts of the ferromagnesian silicates.

Quartz Monzonite corresponds with granite.
Monzonite corresponds with syenite.

Latite is an effusive equivalent, intermediate between the trachytes and andesites.

ANDESITE-DIORITE SERIES.

From the monzonite group to the quartz-diorites the gradation is very slight. These rocks, which mark the persilicic end of the Andesite-Diorite series, are characterized by quartz, with plagioclase as the prevailing feldspar, and with subordinate amounts of ferric minerals. They correspond to granite and rhyolite in the Orthoclase series.

PLUTONIC.	INTERMEDIATE.	ERUPTIVE.
Quartz Diorites. Plutonic or deep seated like granite. *Diorites.* Plutonic equivalent of andesite. A granitoid rock consisting chiefly of plagioclase with either biotite or hornblende, or both. Many diorites carry pyroxenes and shade into gabbros.	*Diorite Porphyries.* Analogous to the quartz porphyries.	*Dacites.* Eruptive like rhyolite. Dacite is a quartz andesite. *Andesites.* Poor or lacking in quartz. They form a group of rocks parallel to the trachytes, and contain plagioclase as a principal constituent, with subordinate biotite, hornblende, and pyroxene.
GABBROS.	**DIABASE.**	**BASALTS.**
Granitoid equivalent of the basalts. Consist mainly of plagioclase and pyroxenes, with various admixtures of other minerals. A large family of rocks, varying from almost entirely plagioclase to near pure *femic rocks*, such as pyroxenites, hornblendites, and peridotites.	Intermediate in texture between the granitoid gabbros and the basalts. Consists chiefly of plagioclase, augite, magnetite, and sometimes olivine.	Contain more femic minerals than andesites. Plagioclase, pyroxene, magnetite, and often olivine are principal constituents. Hornblende rarely occurs.

bottom level of the mine, when assuming a value for the probable extension of ore in depth, due weight must be given to the likelihood or otherwise of exposed ore continuing, and the influence which secondary enrichment has had. In arriving at a decision, the geological factors must be carefully taken into consideration. There are numerous instances where long shoots of ore of good grade have been opened and have extended to a depth of only 200 or 300 ft. below the surface, to be replaced by unprofitable material. The Vivien and Bellevue mines in Western Australia are illustrations of this fact. Many others, of course, could be cited.

CHAPTER VII.

CHURN DRILLING AS APPLIED IN SAMPLING.

There are deposits of copper, iron, lead, and zinc that occur in tabular or massive form, whose greatest dimensions are their superficial area, and which crop out at the surface, or are so near to it that they can be economically penetrated by means of mechanically operated drills, such as diamond drills and churn drills.

For many years the huge deposits of iron ore in the Lake Superior region have been sampled by means of diamond drills. More recently the churn drill, long used for drilling oil wells, has been applied for sampling the disseminated copper ores, from which such an important proportion of the American production is now being obtained. In the zinc and lead districts of Missouri and Kansas, similar methods have been in successful use.

The churn drill has largely supplanted the diamond drill where a vertical hole can be put down, because it is cheaper to operate and gives a more satisfactory sample. The diamond drill hole is 1 to 2 inches in diameter, whereas the churn drill puts down a hole 6 to 12 inches in diameter, so that a much larger sample is obtained. The diamond drill is limited to rock sufficiently resistant to yield a solid core, but on the other hand it has the advantage of being able to drill a hole at any angle.

It is not proposed to discuss diamond drill methods, as these are well understood, and the principal makers will always contract work; besides, the churn drill has come into general use where vertical holes can be put down.

No discussion of mine sampling at the present day would be complete without some discussion of the methods employed in churn-drill sampling of that class of deposits, which were formerly known as low-grade porphyry copper deposits, but are now usually spoken of as disseminated copper deposits. These deposits are commercially important, and though low grade. are worked on a scale hitherto unattempted, so that they now are an important factor in the copper production in the United States. The ores occur as chalcocite derived from the leaching of copper in primary chalcopyrite impregnations in porphyritic and schistose rocks and its redeposition at a lower horizon in the richer form in which it is found. The enriched part usually has a vertical depth of not less than 200 ft. and lies more or less parallel to the surface of the ground. The leached material between it and the surface constitutes what is known as the overburden and sometimes is sufficiently shallow to warrant steam-shovel operations. Beneath the zone of enrichment is unprofitable material, containing the original unaltered sulphides.

Many deposits of this character have been prospected in the United States by the use of churn drills, and enormous tonnages proved by systematic drilling operations. Two types of drill have been thoroughly tested; the Star and Keystone, each possessing particular advantages in certain rocks. The prospective mineralized area is co-ordinated into 200-ft. blocks, that is, divided up checker-board fashion and holes put down at the intersection of coördinates. They are logged and mapped according to the distance from the datum lines, as for example: North 400, East 800, or N4, E8. Barring accident, the hole is continued until the zone of primary sulphides is proved beyond doubt. The holes are sampled, as will be later described, in running five-foot sections, while panning tests are made at every sludging of the hole to determine not only the character of the rock being drilled but also the nature of contained mineral. While running through the barren oxidized overburden, samples for assay are only taken every 20 or 30 ft. By means of these machines, which are modified oil-well drills, a cylindrical section from 12 to 4½ in. diameter, according to depth, is taken, from the surface to the bottom of the hole.

The general method of sampling is much the same in different parts of the country. For most of the following description I am indebted to Lloyd T. Buell of the Miami Copper Co.'s staff and to Frank H. Probert, consulting engineer, Los Angeles, California.

The mechanical operation of obtaining a sample consists of four steps: (1) cutting the sample; (2) removing the cuttings from the hole; (3) reducing the sample to convenient size; (4) drying.

The actual drilling will not be described in detail, but the errors which may affect the sample will be considered later. During the process of drilling, water is poured into the hole as needed to form with the cuttings a thin mud or sludge, and as the tools are being removed from the hole the mud adhering to them is washed back into the hole. The cuttings from five feet of hole constitute one sample, though it is sometimes necessary to remove the tools and clean the hole more than once in advancing that distance. The depth from which a sample is cut is read from measurements marked on the drilling cable as the hole is drilled.

The cuttings are recovered by a bailer which discharges at the surface into a launder, several trips being necessary each time the hole is cleaned. This bailer which lifts the pulp from the bottom of the hole is operated by a dart valve, which prevents the sludge from splashing over the sides of the hole as it is withdrawn. It is only released by the dart striking the bottom of the launder which conveys the pulp to a split sampler. At Miami this sampler, which in effect is a multiple Jones sampler, is made by the company's tinsmith. Part of the discarded pulp from the sampler is caught in a bowl and carefully panned. It is examined by means of a hand lens to record the nature of the rock and minerals passed through. The results are entered in the log. The pulp as it passes through the sampler is divided three times, one-half being rejected each time, so that one-eighth of the material recovered from the hole is reserved as a final sample and collected in a galvanized iron tub, usually about 30 inches in diameter.

The tub is then set over an open fire and the water boiled away and the sample dried and sacked. It is then sent to the laboratory, where, after it has been pulverized and successively reduced until the final pulp is obtained, it is ready for analysis. As a general average, the total amount of dry pulp of each representative five-foot sample is about 40 lb. If the ground is caving, it may happen that the dried sample obtained is too large to go in one sack, in which case at Miami it is coned and quartered, or otherwise divided at the drill to reduce it to convenient size.

The errors peculiar to churn-drill sampling may be classified as follows: 1. Deviation of the hole from the vertical. 2. Separation from the rock at the point of drilling of valuable minerals, resulting in a sample that is too rich. 3. Concentration by gravity of heavy minerals in the bottom of the hole so that they are not recovered when the hole is cleaned, but remain in the hole and follow it down to be recovered at a lower level. 4. Caving of rock from the sides of the hole.

The character of the ground in most of the disseminated copper properties is such that careful drilling will usually insure a vertical hole, deflection being caused by the drill encountering harder rock at an acute angle.

Concentration of minerals in the bottom of the hole by gravity tends to equalize the grade of the samples and to indicate the occurrence of the heavy minerals somewhat below their greatest depth. Theoretically, the extent to which such concentration will take place will depend upon the size, hardness and specific gravity of the mineral particles, and will be minimized by careful cleaning of the hole each time the cuttings are removed, and a knowledge of the manner of occurrence of the ore will prevent a misinterpretation of the results.

Caving is the most obvious cause of error in churn-drill sampling and may entirely vitiate the results. The remedy, however, is simple and effective and consists in lowering casing to the bottom of the hole and proceeding with a smaller bit, repeating the operation three or four times if necessary, each time using smaller casing and a smaller bit. Extensive caving will be indicated to the driller by the feel of the tools while drilling. Comparatively slight falls of rock may be detected in this way, or the caving may be so great as to cause difficulty in extracting the tools. The recovery of more sludge than would be derived from the advance made or the presence in the sludge of material foreign to the ground being cut are evidences of caving. Casing is frequently necessary simply to keep the hole open so that drilling can continue, regardless of the validity of the samples secured. In fact this is practically always necessary in deep holes. Frequently when casing has not been used it is lowered as soon as the ore is reached as a preventive measure, whether any caving has taken place or not.

Permanent records for each hole, known as the log, show the kinds of rock passed through and the limits of each, the important minerals present and the grade and character of the ore, the latter by five-foot sections; also all details of the drilling operations, including mention of caving if it occurs, and the amount and size of casing used. The examination for minerals is made with the aid of a vanning plaque at the drill as each sample is taken from the hole. In this connection it should be noted that excessive heat in drying the sample after the water has been boiled away will result in the oxidation of some of the sulphide minerals, and the sample must be dried slowly if an accurate determination of these minerals from the dried sample is important.

Taking the usual practice in the United States, the hole at the surface is usually from 10½ to 12 inches in diameter and the average diameter at the bottom would be 8 to 8¼ inches, so it will be seen that a large sample is obtained.

According to Mr. Buell the churn-drill results at Miami at the time he wrote had not yet been proved by the underground development work, however in other parts of the country the underground developments have checked the results obtained by churn drilling to a remarkable extent.

In the autumn of 1909 I had the opportunity of seeing the actual results of the sampling of the Ray Consolidated mine. A great many of the drill holes put down were tested by means of rises put up on the line of the hole, half of the hole being carried in one corner of the rise. Individual results vary considerably in some cases, but taken over a series of holes the average values obtained from the churn-drill samples and samples taken in the rises in the ordinary manner check to thousandths of one per cent. The experience then may be stated to be that in sampling deposits of this kind individual results may not be depended upon any more than in the ordinary ore deposits. However, owing to the comparatively even distribution of the valuable mineral in the gangue, samples are not required to be taken with the same frequency as with the usual type of deposits.

The description of the log given above is that used at Miami. Mr. Probert devised a graphic method of presenting all the essential details of churn-drilling work at the Ray Central mine in Arizona. This is shown in the accompanying illustration (Fig. 13). This graphic log presents all the required information at a glance in a much more lucid manner than is possible from a mere inspection of figures, and Mr. Probert deserves great credit for devising it.

So much confidence is placed in the results obtained by churn drilling that it is customary to reckon as proved all ore included within holes 200 ft. or less apart. If holes are set at greater distances, if systematically drilled the included ore is counted as partly proved or probable ore. Already millions of tons have been mined from areas tested in this manner, with results so satisfactory as to warrant the continuance of the practice. That ground sampled in this manner should be accepted as proved ore or

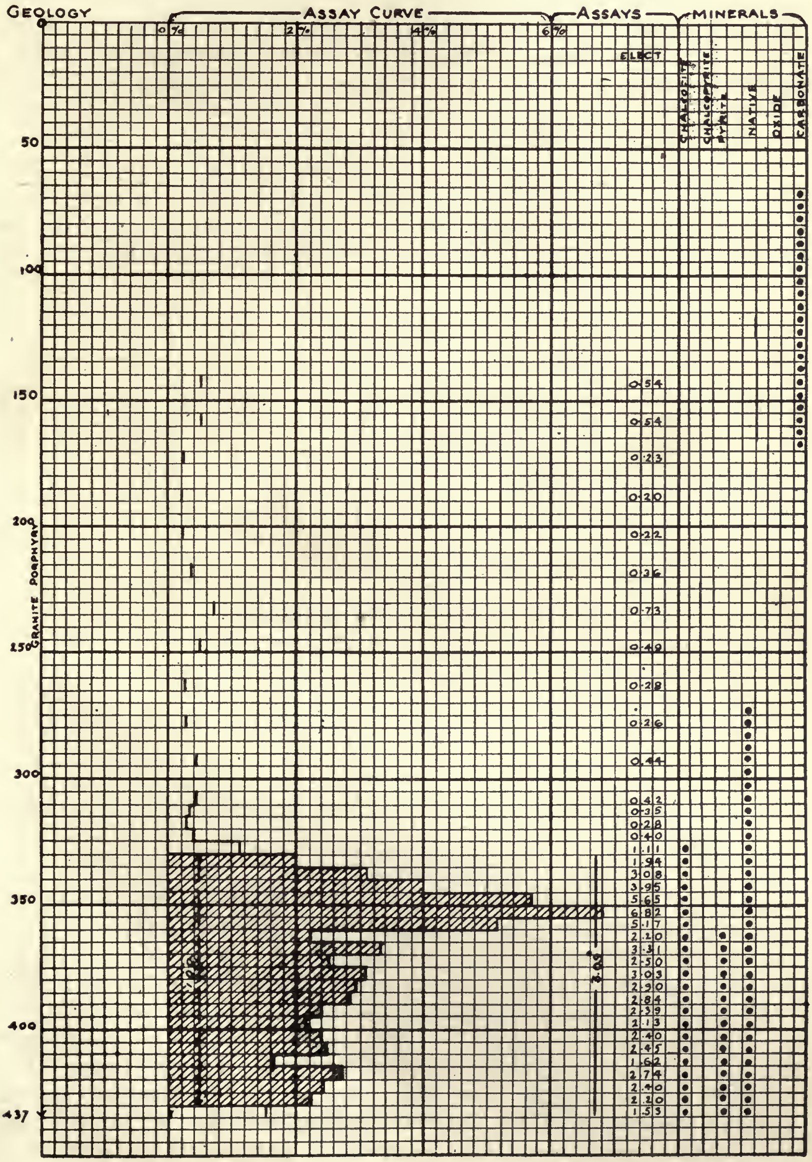

Fig. 13. CHURN-DRILL LOG.

blocked-out ore is a startling demonstration of the fact that experience and judgment are needed by the mine valuer. Combined with the interpretation of the drill-hole assays, a knowledge and comprehension of the geological conditions is essential. Only the comparatively uniform distribution of the valuable mineral over a large area renders it possible to accept the results as sufficiently conclusive for valuation purposes. At the Giroux mine, Ely, Nevada, in correlating the assay results, averages were calculated not only along the coördinates but also along the diagonals, to arrive at a closer check in cases where the results of adjoining holes showed considerable variation in assay values.

In Arizona the average rate of advance in drilling per 12-hour shift is 30 to 40 ft. for the first 500 or 600 ft. This is somewhat better than the results obtained at Ely, Nevada. The costs in both States seems to be the same, namely, $2.25 to $2.50 per foot, to

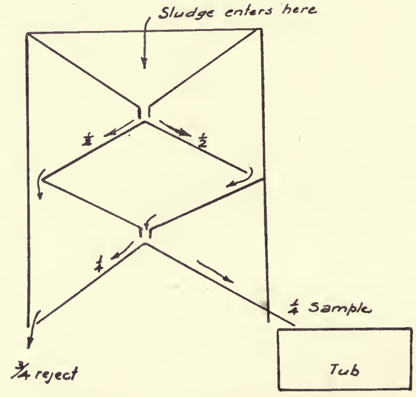

Fig. 14. PLATE'S DEVICE.

which must be added 25c per foot for road building in mountainous country.

Another method of collecting a sample from a churn-drill hole is described as follows by H. R. Plate: The sludge is thrown directly into a wooden box about 6 ft. long, 4 ft. wide and 1 ft. deep and there allowed to settle. The clear water is drawn off by holes bored in the end of the box at different levels. When most of the water has been removed the solid material is sampled with a split shovel. The amount of the sample, which is about one washtub full, is then dried and treated in the ordinary way. The objection to this method of sampling is that a portion of the chalcocite which floats is lost in drawing off the water and the split-shovel method is not so accurate as some of the other methods described.

A device for which Mr. Plate is responsible and used by him at Ely is shown in Fig. 14. It is made of galvanized iron and is about 3 ft. in diameter. The sludge enters at the top and three-quarters of the original amount is rejected, the remainder being caught in the tub and treated in the manner already described.

H. R. Krumb devised an apparatus that in effect is a double riffle sample, one placed above the other. The rejected portion flows out at the ends of the troughs and the quarter that goes through is caught below.

CHAPTER VIII.

HANDLING OF SAMPLES.

We have now arrived at a stage in the discussion at which it may be assumed that the sample has been properly taken, and that the broken mineral is to be sacked and removed for assay. The following suggestions for handling the sample are for the guidance of the younger men, although it is hoped that even the older men may find some points of interest.

Numbering the Sample.—The samples as taken from consecutive intervals should be given consecutive numbers. Some engineers, in order to mislead the assayer, do not use consecutive numbers. This results in a great deal of extra work when sorting the assay returns and thereby introduces another possibility of error.

The most simple and flexible system of enumeration is to give each part of the mine a different basic or serial number, somewhat after the so-called decimal system. For instance, along No. 1 Level North, we may start off one series as numbers 1, 2, 3, etc.; then 101, 102, 103, etc., to represent samples, say, from a particular stope; 201, 203, 204, etc., say, from a certain series of winzes or rises and so on through the mine. This facilitates the work of the examining engineer and tends to prevent mistakes, because in case of error the samples are so easily traceable.

The fractional samples in each particular interval are marked with the interval number and going toward the footwall are denominated *F1*, *F2*, *F3*, etc., and toward the hanging wall *H1*, *H2*, *H3*, etc. If there are only two fractional samples from the foot and hanging wall side respectively they may be marked *N. H.* and *N. F.;* or *N., NF.;* or *N,* or *N. H.,* in each case *N* standing for the sample number.

It will be seen that this allows of the introduction of additional samples in any portion of the mine, without making an enormous jump in the consecutive numbers. It may be argued that this system, although flexible, is cumbersome on account of the long numbers that must be used where many parts of the mine are given each a separate serial number. In my opinion it is preferable to write a long number on a paper which when seen immediately indicates the locality of the sample, than it is to have lower numbers and to be under the obligation of searching sampling records, in order to determine its exact position. For the same reason the keeping of the assay records is greatly simplified as the number indicates its position in the record book.

Some engineers use numbered metallic disks, but these are objectionable because they are bulky and because there is no flexibility in such a system. The use of metallic disks necessitates giving a separate number to each fraction, and in this way the nomenclature is so complicated that it is practically impossible for an engineer looking at the sample number afterwards to tell its position underground. The author has used linen labels for a number of years with great success, and has never found a sample number mutilated to such an extent that there was any difficulty in recognizing it. Besides the flexibility in numbering that this permits, the portability of these labels strongly recommends them. In the absence of linen labels, strong bond paper or drawing paper should be cut to suitable size.

Tags made of wood are recommended by some engineers. My own experience is that these are very unsatisfactory. In the first place, they are troublesome to make, they are bulky, the number is liable to be obliterated in transport, or the tag may even be broken and crushed to an extent to make it difficult to read the number, besides contaminating the sample with the slivers of wood.

Some engineers recommend putting the sample number on the bag for identification and in order to prevent outsiders learning of the identity of a sample it has been suggested by one man to put the number inside the bag, close to the string. Numbering the bag is a nuisance as it rarely arises that the number of a sample need be known after it is sacked, except where two bags are used for one sample, in which case they should be tied together.

If there was any valid reason for putting the sample number on the bag, there certainly is no excuse for concealing it by placing it on the inner side. This is an admission of bad practice or carelessness. No sample should ever be left about, where it can be seen by those who could take advantage of its identity. A sample from the time it is taken to the time it is assayed, should be safeguarded from interference by outsiders.

Description of Sample. After the sample has been cut, the sampler's first duty should be to make a comprehensive description of the sample in his notebook. The following, which is more or less self-explanatory, is a copy of a portion of the author's sampling record on one property:

Zig Zag Workings.* (See mine plan). 1st long level—According to the plan this is about midway between the Old 'L' and No. 2 levels. At Rise 670 floor is 12 ft. 6 in. above the floor of intermediate, or about 26 ft. 6 in. above Old 'L' floor.

761. In west face of 26-ft. level—This is 10 ft. west of west side of Rise 670. 51 in. nice looking soft gossany ore containing Mn and quartz and some country rock. Neither wall reached.

762. In drive of 26-ft. level, just above west side of Rise 670—30 in. of mixed ore from footwall. The bottom half of this is almost completely replaced country with some quartz, the remainder soft ore with quartz, and contains manganese.

H1.—39-in. gossany ore, mostly soft, but containing a large amount of highly altered country rock near hanging side of cut, whereas quartz and manganese predominate in other part of sample. Hanging wall not reached.

763. 10 ft. east of 762 in 26-ft. level—16 in. of quartz on footwall.

H1.—20 in. of gossany ore, containing considerable manganese.

H2.—21 in. of quartz ore, with small amount manganese.

H3.—36-in. gossany quartz, with manganese and clay. Hanging wall not reached.

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*Name given to workings on account of their appearance on mine plan.

764. 10 ft. east of 763—34 in. of soft gossany ore, containing manganese, clay and quartz, taken to back of drive. Footwall not reached.

764H.—39 in., same ore, across back of drive. Hanging wall not reached.

765F.—10 ft. east of 764, 15 in. of footwall stuff, apparently mostly country.

765.—26 in. of hard gossany white quartz.

765H.—42 in. of mixed gossany ore (quartz, clay, and manganese, with nodules of country). On hanging side 6-in. manganese. Hanging wall not reached.

Resamples for Checking Results.

138R. 10 ft. east of 137—Footwall here is of red rock. 36 in. of hard gossany quartz.

138RH.—60 in. of soft ore containing some quartz, and considerable manganese. Hanging wall getting quite smooth, clayey material next it.

140R. 10 ft. east of 139, opposite cross-cut 990 East—44 in. of hard gossany quartz, measured from retaining wall. Footwall not reached.

140RH.—25-in. soft ore containing some manganese, and a little hard quartz.

142R.—24 in. of soft gossany ore with manganese on hanging side. Below this sample on foot is altered decomposed country, containing some quartz, which was not sampled.

144R. 8 ft. east of 143R in the timbers—39 in. of gossany ore containing some quartz, but mostly clay and manganese ore, to hanging wall.

It will be noted that in the resamples quoted only alternate numbers are given, the odd numbered samples were taken by one of my assistants. This is to emphasize the advisability of having two men sample in company, each taking alternate intervals. In this way if there is any error due to personal equation it will be discovered; besides, if the responsible engineer be one of the two, he is enabled personally to direct where and how the samples are to be taken and incidentally watch his assistant's work. In resampling it is, of course, not necessary to describe the sample fully as this has already been done. Resamples should generally be taken along the same cut as the original.

The descriptions, such as here given, may mean nothing to one unfamiliar with the workings, but are intended as a guide to the engineer himself in case of unexpected results. With copper or lead mines, it is useful to make a guess at the assay value. A small lump of chalcocite placed in a sack of low-grade copper ore by an interested party will materially raise the grade. If the value is estimated and set down in the notebook, the salting may be detected.

The sample number should be plainly noted in the margin of the page, followed by its distance from the last sample, or the nearest survey station. In addition it should, if possible, be tied to any of the other mine workings, such as a cross-cut, rise, or winze. This facilitates locating the sample on the mine plan afterwards. The width sampled should be measured and noted, followed by a description of the character of material. For instance, if a high result was obtained from a sample containing much country rock, it may be taken for granted the result is incorrect.

This procedure serves the purpose not only of impressing on the engineer's mind the characteristics of the orebody, but may enable him to trace causes of erratic assay values, as well as indicating in what class of material the values are contained.

Sacking the Sample.—Assuming that the cuttings of ore comprising the sample are in a sample bucket, it becomes necessary to transfer the sample to one of the small bags provided for the purpose. It has been recommended that two sizes of bags should be provided and it is strongly urged that a bag large enough to take the entire sample be used. Dividing the sample into two or more bags tends to error in the assay office.

Where a gold mine is being sampled, the sample sacks should be turned inside out and beaten, in order to shake out any grains of gold dust that may 'inadvertently' have been put there by interested parties. Before the sample is put in the sack, it must be again turned, so that the seam is inside. In order to transfer the sample to the sack, one of the smaller sheets of canvas is laid on the ground and by means of the hands, or by pouring from the bucket, the sack is filled. Care must be taken that none of the particles are lost.

Brittle minerals generally contain the metals of highest value and because of their brittleness powder easily. For that reason, the last portion of the dust remaining in the bucket or what may overflow onto the canvas, must be carefully gathered and added to the sample. The inside of the bucket should be beaten with the hand or other object to get out these last portions.

Some engineers may say that the discarding of the dust acts as a factor of safety, but this is bad practice. A loss of any portion of the sample renders the result inaccurate and if the engineer desires to provide a factor of safety, he should do so with his eyes open and not leave it to chance.

The ore being in the sack, its number should be marked on one of the small linen baggage labels with an indelible pencil. If there are two sacks composing one sample, the two labels could be written out together and, to avoid error in the assay office, each one is marked 'Two Sacks,' and the two sacks tied together after they are sealed.

Tying and Sealing.—This sounds like a comparatively unimportant operation, yet there are a few precautions which it is advisable to indicate, the neglect of which may in some instances be the cause of erroneous results.

The ore should be well shaken down in the sack, so that it will occupy the minimum space, and the linen label placed on top. The mouth of the bag is then firmly held together and the string tied below the hand as close to the ore as possible. This operation can better be done by two people than one. The sack should be held facing the person tying it, and taking a double turn of the string around the neck, a single knot is tied on the farther side of the sack and drawn tightly together. The string is then passed around to the front and a firm double knot tied as near the centre of the sack as possible, care being used not to tie a granny's knot. The object of the first knot is to prevent the string loosening while tying the second knot; besides, it is essential that the mouth of the bag be tightly closed to prevent any possible loss of fine material during transportation, especially when samples are shipped long distances for assay. This being done, the sack is sealed with a clear impression of the seal on this last knot, care being taken that the sack is not burned during the operation. Some engineers use a sack already supplied with strings, which are sewn on. As these are always near the mouth of the sack, when the sack is not full, there is not only danger that the loose ore will so mutilate the label as to make the number illegible, which, of course, means a resample, but tampering with the sample is rendered more easy.

It is very difficult to tamper with the sample that is sacked and tied in the manner described, as any addition to the ore is impossible without showing evidences of tampering. Adding gold dust to the mouth of the bag is a hazardous operation as the chances are that none of the gold dust will ever enter the sample, but will be shaken out before the sack is opened or be discovered in the assay office.

Of course, with the precautions that are here recommended as to the safe-guarding of samples, there is not much likelihood of their being molested by others after they are once sacked, but too many safeguards cannot be thrown around the samples, as on them the whole final results depend. The engineer must not only protect the interests of those by whom he is employed, but he must bear in mind that his own reputation is at stake, and that one serious error in mine valuing is apt to damn him forever. One thing that financiers cannot forgive in an engineer is the loss of money through that engineer's advice. The sample once sealed should be put in the leather mail sack, and when the samples are taken from the mine, if it is not done before, the sack should be locked.

Before proceeding to take the next sample, sample buckets that have been used should be turned inside out, and if the ore be dry, should be thoroughly brushed with a scrubbing brush. In wet mines the scrubbing brush should still be used, but with the addition of water to wash off any remaining portion of the previous sample. The canvas sheets used should also be thoroughly cleaned before using again.

A distinctively colored string, as sometimes recommended, is no additional safeguard, as the sealing of a sack effectually prevents tampering with the string, besides which, colored string is not always obtainable.

CHAPTER IX.

PREPARATION OF SAMPLES FOR ASSAY.

This perhaps belongs more in the realm of the assayer than the engineer, but in practically all cases where an engineer is called upon to conduct a mining examination, he must watch the sample all the way through, even to the assaying. It would be useless to adopt all the precautions one usually does in taking the sample and the prevention of salting underground, if in the end the sample is turned over to some irresponsible person for crushing and assaying. It is very easy to accurately salt a sample, if one has the handling of it in the assay office, because there weights can be accurately determined and the sample scientifically salted to any desired extent. When an operating mine with a complete assay plant is being examined, this problem is simplified, because the engineer usually is given possession of the assay office and such apparatus as it may contain.

It may be stated without cavil that there are comparatively few mines that engineers are called upon to inspect, fitted with crushing apparatus of sufficient capacity to permit of rapid handling of the large number of samples usually taken. It then devolves on the engineer to make some arrangement to facilitate his work, and often considerable ingenuity must be employed to prevent the loss of an excessive amount of time. This may be illustrated by one or two instances from practical experience.

In one examination where over 1,000 samples were taken by an engineer and two assistants, the option period was limited and the assay laboratory was fitted with two small hand jaw crushers, one bucking board, and one pestle and mortar as the sole crushing appliances. Another bucking board was obtained from a mine nearly 150 miles away by road, and an additional pestle and mortar secured in the neighborhood. With eight native laborers in this crushing room, working long hours, about 30 samples per day were prepared for assay, the engineer himself, or one of his assistants, being in constant attendance watching the operations and doing all the quartering. Fortunately the laboratory was provided with two Jones dividers, which greatly facilitated the reduction of the samples.

Some years ago I was unexpectedly called upon to examine an ancient silver-lead mine in Burma. In the course of the work some 500 samples of lead slag carrying about 50% lead were taken. There were no crushing appliances whatsoever on the property and it was impossible to get any in the country. The only thing to do was to make the best of the conditions that existed and to prepare the samples for assay by hand. These were obtained from test pits put down in slag piles and some of them weighed as much as 200 lb. The following *modus operandi* was employed: Large canvases were laid on the ground and as many coolies as possible placed around the canvas, each provided with a hammer and a large rock, to serve as a mortar block. The lumps of slag were crushed until the whole sample would pass about a ¼-in. mesh screen. It was then passed over a riffle divider, with which I had fortunately provided myself. This divider was set up over an empty kerosene box and supported at the corners by means of four long nails. By subsequent recrushing and dividing, the sample was brought down to proper weight for fine crushing. Two gangs of men were employed in charge of an intelligent Burman under my direction and considerable speed was attained. The results were accurate, as the subsequent smelting operations demonstrated.

The following may be taken as a general method of preparing mine samples for assay. In some cases the ore may be so dry as not to require any preliminary drying. On the other hand, accurate results in reducing the sample cannot be obtained if the ore is moist. The first step then is to dry the ore, if it be necessary. Here again we meet with a problem. If the ore has a tendency to be sticky, great difficulty may be experienced in thoroughly drying the sample, it occasionally being necessary to leave such a sample on a hot plate for six or eight hours, or more. In the first illustration quoted in this chapter, the ore was clayey and wet. Fortunately at a previous examination of the mine, a hot plate had been provided by using a sheet of ¼-in. iron, set upon brick walls fitted with a door for firing in front and connected with a chimney, so that a hot fire could be maintained.

No doubt theory demands that in order to secure accurate results, moisture should be evaporated from a sample at a temperature only slightly above the boiling point of water. Had such a procedure been adopted in this case, it would have taken months to dry the samples. As a matter of practice, it was found that the only way to get results from some of the samples was to have the plate heated to redness. The water could scarcely be driven out of the clay, except by making brick of it. It can be readily understood, that unless the clay had been heated to a sufficient extent to make it hard, it would not have gone through the crushers and any accurate division of the sample would have been impossible. This is cited as a typical case where the engineer must use his judgment and not blindly follow theory. The objection to heating a sample to the extent stated can only be that some of the volatile elements will be driven off and perhaps with an attendant loss of some of the precious metals. But this is inevitable and if such a thing were to occur, the error could only be on the side of safety. This sort of factor has been deprecated in these pages, but in this case there is no remedy.

Russian iron pans were used to hold the sample while drying, each pan having the sample number marked on it with chalk. If there are two or more pans for one sample, it should be so noted on each of the pans. The original labels found in the sacks are put on one side and at the end of the day checked up with the finished samples, in order to immediately trace any errors. Mistakes are bound to happen when one man has to follow a number of samples through the various operations that are carried out by unintelligent labor. When the sample is dried, it is put on one side to cool and relabeled by a slip of paper bearing its number, preparatory to carrying it through the next series of operations.

Occasionally erratic results are quoted by engineers, but these are often due to mistakes. A case discussed in Mr. Rickard's book* showed that two halves of an original sample gave widely varying results, because the reduction of a wet, sticky sample was undertaken underground, without any drying. The sample contained wire gold and was mixed on a canvas. Accurate results could have been obtained only by accident. A sample containing any clayey material should be thoroughly dried before any division is attempted. A sample of low-grade quartz may perhaps be divided wet, but if there are any valuable sulphides or metallics, even with a clean quartz gangue, the sample should be dried first.

Where proper mechanical crushing appliances are available, a sample can be pulped in a very few minutes. Various types of crushing apparatus are on the market, some better than others, but all more or less well known. The selection of such apparatus rarely falls to the examining engineer and, therefore, can be dismissed without much comment.

Occasionally, however, in going to foreign parts, the examining engineer must provide himself with portable outfit. Of the hand crushers, the one with fly-wheels is more efficient and preferable to the lever type. A mechanical grinding machine operated by hand is a useful adjunct. It is a great time saver over the pestle and mortar, as jaw crushers cannot be depended on for fine crushing and there must be some intermediate apparatus between this and the bucking board. Only one caution is necessary in the selection of apparatus, and that is, care must be exercised that the machine be strong, as near fool-proof as possible, and that parts are easily accessible for cleaning.

After the sample has been thoroughly dried and relabeled it must be crushed to small size, before being divided. In the instances previously cited, where there were two small jaw crushers, one of them was set for coarse crushing and the second for fine. In the latter the ore was reduced to pass a quarter-inch screen, although nearly all was much finer. It is well known that if material is crushed to pass a given mesh, then probably 90% would be much finer, so that a great deal of time can be saved by crushing to a coarser mesh than desired, screening out the oversize and recrushing. The fineness to which a sample must be crushed for division depends altogether upon the bulk of the sample. A number of engineers have contributed to our knowledge of the subject and these articles are easily accessible. A paper by D. W. Brunton, published in Vol. 49 of the Transactions of the American Institute of Mining Engineers, is a classic and should be read by every mining engineer. Some of the works on assaying also treat of this subject.

*'Sampling and Estimation of Ore,' page 151 et seq.

Speaking generally, the fineness to which the ore should be crushed before further division, depends to a considerable extent upon the character of the material that is being sampled. The same rule applies in this case, as does to sampling the ore underground, namely: The more homogeneous the material, the more evenly distributed the valuable mineral, the lower the monetary value of that mineral, then the less care is required. On the other hand, in the case of irregularly distributed gold with high-grade brittle sulphides or free gold, great care must be taken, more especially if the ore is not of high assay value, because a small proportion of the valuable mineral going into one half of the sample or the other may seriously affect the result. Where coarse gold occurs the problem becomes exceedingly difficult and serious; errors are liable to occur and the greatest care must be exercised in manipulating such samples during the reduction to the final pulp.

It will be seen that no general rule can be laid down. The engineer must use his judgment in the matter. In this connection, it may be pointed out, that as an extra precaution, it is advisable to select a few samples, carry both halves of the original sample through to a final pulp and assay both original and duplicate. This will serve as a check on the operations and, if no serious differences are noted, the engineer will have the assurance of knowing that the work has been properly done. On the other hand, if there be serious differences in the assay values of the original and duplicate, it will be a warning to revise the method employed.

I think there is a tendency on the part of far too many engineers to make a division of the original sample before it has been crushed to a sufficiently small size. Probably the main reason for this is the amount of time and labor it requires to do this work.

Division of Sample.—After the sample has been crushed to the proper fineness, it must be divided, if a mechanical divider is at hand; if not, the sample must be quartered in the ordinary primitive way. When a mechanical divider is used the operator must see that the ore is distributed evenly over the scoop, and emptied over the divider in an even stream, care being taken not to fill the divider to overflowing. Sometimes the sample, if large, can be divided a second time without further crushing, but in this the engineer must be guided by his experience, always bearing in mind that the greater the fineness, the more accurate the result. In some cases it may be advisable to crush the whole remaining material to pass a 10-mesh screen before making the division to secure the weight required for the final pulp.

A discussion of the methods of preparing a sample for assay will not be complete without some brief reference to the well known method of quartering. Of course, most books on assaying deal with this subject more or less in extenso, and I only wish to briefly refer to it to round up the present discussion.

An engineer may find himself in the field without having provided mechanical dividers of any kind, yet large samples must be reduced in bulk in order to permit of economical transportation.

Without mechanical dividers, the only thing that is left to be done is to quarter the sample on a canvas sheet or on an ordinary oil cloth, what is known in England as American cloth, which can be bought at a general merchandise store, practically in any part of the world. The only other accessories that are required are a stiff brush, such as a scrub brush, whisk broom or horse brush, and the usual implements that are found on any mine.

The theory underlying quartering is the same as that underlying any other way of dividing a sample, namely, to take a fraction of the total bulk and have such fraction represent the original, i.e., have the same assay value. The method of procedure is as follows:

The ore is crushed to a sufficient degree of fineness, the size of the largest particles depending on the size of the sample and the character of the mineralization. This crushing down is done with a hammer, using a worn-out mortar block of a stamp battery, any heavy piece of iron, or even a large stone as an anvil on which to break the lumps of ore. When the ore has been crushed sufficiently fine, it is thoroughly mixed on a canvas sheet. This operation is a simple one, if the proper method be followed. What is required is a rolling motion, so as to get a thorough mixture. If the corner of the canvas is simply lifted, the ore may slide down the inclined surface and no mixing take place. With a large sheet, one end is allowed to remain on the ground; one corner of the opposite end is then taken and by a twisting motion the ore *rolled* over and over on the sheet, so that the desired mixing effect takes place. The same thing is done alternately from each corner and the same action is repeated from the opposite end when the ore has rolled up far enough. When the ore has been sufficiently mixed it is coned on the canvas sheet, that is, a small quantity is poured into a conical heap in the centre and successive amounts added to it, care being exercised that the stream of ore, as it leaves the shovel or scoop, falls as near as possible onto the point of the cone, the intention being to distribute the flowing stream of ore equally on all sides. This cone is flattened by spreading the ore from the centre to the outer edges again, using a circular movement with a view of continuing the mixing. For spreading the ore, a shovel or scoop is used. The shovel is held upright and by a series of vertical reciprocating movements, at the same time moving the shovel in a circular path, the cone is flattened down. The top is smoothed off, working from the centre towards the edges and then divided as nearly as possible into four equal quarters by means of a flat strip of wood or a shovel. Two opposite quarters are rejected. The ore within the two diameters marking them is carefully removed and the dust belonging to these quarters is carefully gathered with a brush or small broom and also rejected. With brittle sulphides the dust may show the higher assay values and consequently an undue proportion must not be allowed to remain with the part of the sample which is retained, as the result may thereby be vitiated. Successive quarterings are carried out with crushing at each stage, dependent on the size of the sample, until it is small enough for the assay laboratory.

Fineness of Crushing.—The fineness to which ore must be crushed depends very largely on the character of the minerals and the value of the richest mineral in the ore.

A valuable paper by D. W. Brunton on this phase of the subject appears in Vol. 25 of the Transactions of the American Institute of Mining Engineers. Mr. Brunton describes a series of experiments which were undertaken to determine the fineness to which crushing must be carried in sampling gold and silver ores in order to obtain results within an allowable limit of error.

In a résumé of Mr. Brunton's calculations, which will serve a useful purpose if they do no more than call attention to the importance of this subject, he states:

"The experiments and calculations described, and a general consideration of the subject indicate that the size to which ore must be crushed for sampling in order to come within an allowable limit of error will depend upon: (1) The weight or bulk which the sample is to have. Evidently the smaller the sample the finer the material must be crushed. (2) The relative proportion between the value of the richest mineral and the average value of the ore. If the average grade of the ore is high, in comparison with the grade of the richest mineral, a particle of richest mineral of a given size and value will have less percentage effect on the sample than the same particle would have on the same amount of lower grade ore; therefore, other conditions being the same with high-grade ores, we may crush more coarsely than with low-grade ones, and still keep within the same percentage of error; while if the richest mineral is of comparatively high grade a particle of it of given size will have a greater effect on the sample than if it is of low grade, and this will necessitate finer crushing. (3) The specific gravity of the richest mineral. The higher the specific gravity of the richest mineral, the greater the value contained in a particle of given size and grade, and hence the greater the influence of such particle on the sample; from which follows the necessity of keeping down the size of the largest particles by finer crushing than is required when the richest mineral is of lower specific gravity. (4) The number of particles of richest mineral which are likely to be in excess or deficit in the sample is evidently an important factor; a liability to a large number necessitating especially fine crushing. But such liability can result only from imperfect mixing, and, for material mixed with average thoroughness, this number must be small.

"The relative proportion by weight or bulk of the richest mineral and the low-grade or average ore will not of itself affect the size to which the ore must be crushed. . . . . assuming a lot of ore properly crushed and mixed, the principal effect which a large proportion of richest mineral has is to increase the proportion of maximum-sized particles of richest mineral, with reference to the whole number of particles composing the lot, or, in other words, to increase the probability of the occurrence of maximum variations. But the limit of the magnitude of these variations is just the same as when only a small proportion of the richest material is present.

"Inasmuch as the effect of a particle in excess or deficit in a sample depends only on its weight and value, as compared with those of the sample itself, it is evident that the size or weight of the lot of ore from which a sample is taken, or, in other words, the proportion of the lot of ore taken as a sample, does not directly enter into the question of determining the maximum size which the ore may have.

"The results of the investigations recorded in this paper show how absolutely necessary it is that ore samples should be re-crushed after each successive 'cutting down,' so that, as the sample diminishes in weight, there may be a nearly constant ratio between the weight of the sample and that of the largest particle of ore contained therein."

These quotations, above all else, indicate the necessity of crushing the ore as much as possible before any division of the sample takes place.

Preparation of the Pulp.—Ordinarily, only a bucking board is available for grinding, although there are machines that will grind ore to 100 mesh with great facility. A bucking board and elbow grease are what most engineers encounter.

All books on assaying point out the necessity of passing every portion of the pulp through the screen and carefully inspecting the final portion for the detection of metallic particles and state that the metallics should be collected, weighed, and assayed separately. Where the bucking is done by an intelligent person, there is greater security that this final operation will be carried through in the manner that it should be. Where one is depending on unskilled or 'native' labor, the engineer is confronted with the fact that these people neither appreciate the responsibility connected with their work, nor do they understand the reasons therefor, and unless the engineer is particularly careful, he may find either that the last portion of the sample that does not easily go through the screen will be thrown away or it will simply be added to the pulp without being reduced to the proper mesh. This, needless to say, is bad practice and may lead to erratic assay results. In one case, on taking a day's work and rescreening the pulps, it was found that about 25% of them contained some oversize, representing this final difficultly crushed material.

Many people recommend grinding all samples to 100 mesh. In most cases a coarser mesh may be employed without inaccuracy. Here again one must be guided by the character of the material that is being sampled; 100 mesh should be used when the ore is a high-grade gold ore. Low-grade copper ore carrying 2 to 4% of copper or thereabouts and little or no precious metal values can be put through even 60 or 70 mesh. Speaking generally, it will be found that an 80-mesh screen will answer all ordinary circumstances. It must be borne in mind that the finer the mesh, the more time required to buck down the sample.

Final Sample.—When the pulp has been crushed to a sufficient degree of fineness, it should be put in a sample envelope with a flexible metallic strip, which is used for securely closing the envelope. These envelopes are sold by all American dealers in assay supplies, but I believe are not manufactured in Great Britain. A substitute can be obtained, specially made by a stationer, in which the metallic strip is replaced by a strip of paper, but this is not altogether secure. Canvas bags are too bulky for this purpose and bags of muslin or other light material permit of some of the finer dust sifting out through the mesh of the cloth. Paper envelopes have the great advantage not only that they are perfectly secure, retaining none of the pulp when emptied prior to weighing up a charge, but permit of the sample number being plainly marked on the outside, in such a manner that any sample can be easily picked out when required for re-assay or other purpose.

In addition to marking the envelope on the outside, the slip of paper bearing the sample number, which has been carried through the successive crushing operations, should be placed in the envelope to make assurance doubly sure. It is perhaps needless to point out that the marks on the envelope and the paper should be compared whenever the sample is used.

CHAPTER X.

ASSAYING OF SAMPLES.

I seriously hesitate to enter upon this field, as I do not profess to be expert in the art, but my excuse for contributing a few remarks on the subject is that in talking the matter over with a number of young engineers, I find there seems to be a lack of information regarding certain points, a knowledge of which greatly facilitates the speed with which this portion of the work can be done. No doubt all that follows can be found in text-books on the subject, although probably not in the form here set out.

The speed with which assaying can be done depends on the system employed. In American mines and smelting works, according to the usual method, neither crucible nor other object used in assaying is marked with a distinctive number. The samples are set out in any desired order or succession and their numbers noted in the assay record book in the same order. They are then taken one by one in succession and weighed out for assay and the same order maintained in all the successive operations. Until the final result is reached, there is no way to distinguish one assay from another, without counting back.

To give a concrete illustration, let us assume that sample numbers 1 to N inclusive, in their respective envelopes, are placed in order alongside the pulp balance and that a muffle furnace, by far the cleanest and most rapid method for fire assaying, is to be employed. Further that this muffle will take nine crucibles at a charge. Nine crucibles on a board covered with asbestos board and shaped as shown in Figure 15 are placed beside the assayer and so arranged that No. 1 crucible to take No. 1 sample is in the upper left-hand corner, No. 2 to the right of it and so on, working from left to right, so that No. 9 is the last crucible in the lower right-hand corner. When ready for charging into the furnace sample No. 9, the crucible in the lower right-hand corner should be placed in the muffle in the back left-hand corner, and the others following, placing them from left to right, until No. 1, the last to go in, is in the front right-hand corner. See Fig. 16.

When the fusion is completed, No. 1 sample, which is in the front right-hand corner, is taken out first and poured into a mould, so that it comes in the upper left-hand corner. Crucible moulds usually having three or six moulds are placed side by side, and the succeeding samples are poured into the successive holes of the moulds, until one mould is filled, when the next one is started and filled in a similar manner (see Fig. 17). As soon as the buttons have cooled sufficiently, each should be separated from the slag, cubed in the ordinary manner, and placed in a button tray. This is made of sheet iron, divided up into square compartments of any number, 25 is a convenient size, each compartment being perhaps an inch and a half square (see Fig. 18). No. 1 sample again comes into the upper left-hand corner, and working from left to right the succeeding ones follow.

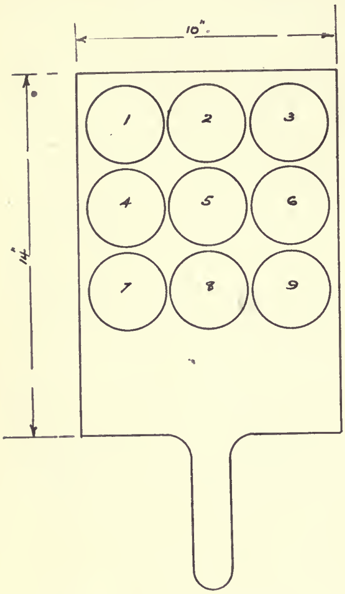

Fig. 15. CRUCIBLES READY FOR CHARGING INTO MUFFLE.

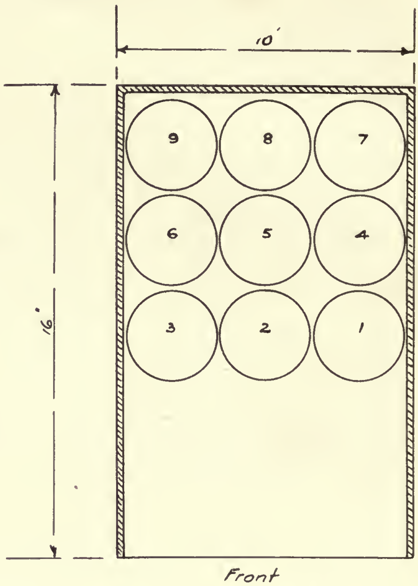

Fig. 16. CRUCIBLES IN MUFFLE.

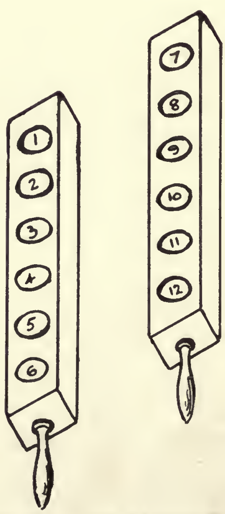

Fig. 17. METHOD OF POURING FUSIONS.

When it comes to cupellation, the cupels are placed on the asbestos-covered board, which has been previously used for the crucibles (see Fig. 19), with No. 1 cupel in the upper left-hand corner and the last one in the lower right-hand corner. The last sample being placed first in the muffle (see Fig. 20) and the others following, working from left to right, until No. 1 occupies the

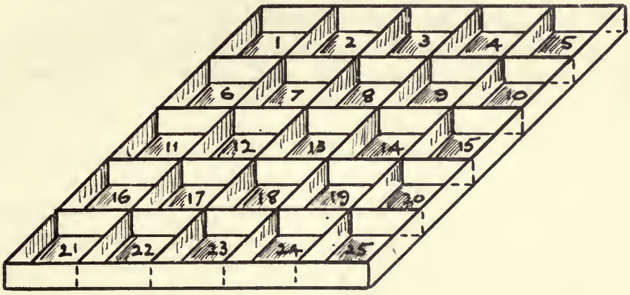

Fig. 18. LEAD BUTTON TRAY.

front right-hand corner of the muffle in the same manner as when charging the crucibles. Taken out in the inverted order, No. 1 again occupies its place in the upper left-hand corner and so on. The same system is followed with the gold and silver beads, and is carried right through the parting operations and annealing of the gold and final weighing. In recording the results in the assay

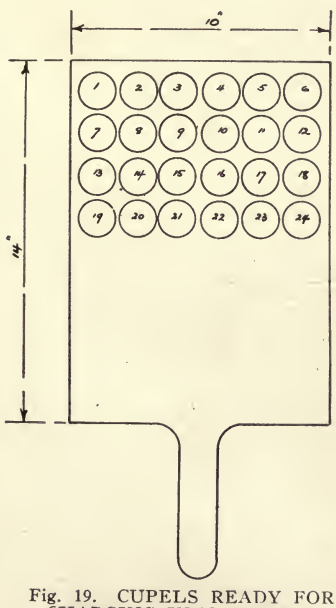

Fig. 19. CUPELS READY FOR CHARGING INTO MUFFLE.

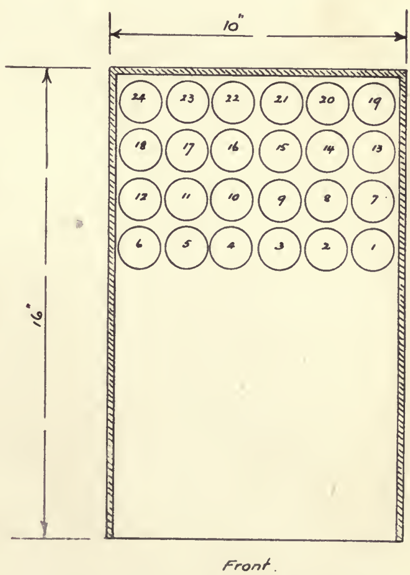

Fig 20. CUPELS IN MUFFLE.

record-book, No. 1 result is put down opposite the first number in the book, whatever its actual serial number may be, and so on in succession.

This method may appear complicated at first glance and liable to cause error, but it is an enormous time saver and if the work is intelligently carried out, there is less liability to error and certainly less loss of time than when every crucible, button, cupel, and annealing cup is marked with its distinctive number.

It is used throughout the United States and in other parts of the world in large mines and works, and where the element of time is so important, as usually is the case in mine inspection, this fact should be a sufficient recommendation for its use. The same system of placing the utensils should be used throughout all the operations. With two men in an assay office working together, anywhere from 100 to 150 samples may be in process and where a large number of samples have been taken, it is practically impossible to put a number of three or four figures on a crucible with any assurance that it can be recognized after a fusion has taken place and some of the slag crept over the top and down the sides, glossing the clay and thus obliterating the number.

PART II.
MINE VALUING.

CHAPTER XI.

ESTIMATION OF ORE.

Part I has been taken up with a discussion of the methods of sampling. We next have to consider the interpretation of the assays of the samples, giving due weight to modifying geological and economic conditions.

It may be taken for granted that the engineer is by this time in possession of a plan and section of the underground workings. If none existed previous to his visit to the property, then one of his first duties should be to prepare them, not only for his guidance during the preliminary inspection and subsequent sampling operations, but also as being necessary for the preparation of an assay plan. An assay plan is a graphic representation of the manner in which the valuable minerals occur and serves as a guide in determining the assay value and tonnage of ore in sight.

Faults, intrusions, changes in the formation, and any other geological data which may seem to have any bearing, should be noted on the mine plans, as an additional aid in sizing up the ore deposit. It may be stated as a broad fact, that the same class of geological phenomena usually occur throughout the same ore deposit; for instance, where faulting has taken place it will generally be found that the orebody is displaced in the same direction in each successive instance, so that if a mistake has been made in the development work and a particular face has passed out of the ore beyond a fault plane, the location of these faults on the plans will show in which direction a continuation of the orebody may be looked for. This cannot be taken as a hard and fast rule, because often reverse faults exist and then the continuation of the orebody must be looked for in the opposite direction.

Before proceeding to plat the assay results on the mine plan, the engineer must do a considerable amount of preliminary work by way of correlating the assays. The orebody has been sampled in fractions and intervals, not only on account of irregularity in the exposures but also because more than one character of ore may be exposed at particular intervals.

Calculation of Averages.—The calculations should be carried out in a book or books, especially ruled off for that purpose and divided into sections for each portion of the mine; that is to say, to correspond to each serial number. These books may be labeled 'Ore Reserve Calculations.' I find a journal ruled foolscap-size book as useful as any. The accompanying diagram shows the ruling and headings—the last two columns are obtained from calculations.

The sampling record from the underground note books is transferred to the left-hand page and the results of the assays taken from the assay office records noted on the right-hand page.

DIAGRAM 1

1	2	3	4	5	6	7	8
Sample No.	DESCRIPTION	Width Sampled, inches	Gold	Silver	Total Au and Ag	Inch by Dollars	Assay value of Interval
101	3 ft. west of survey plug No. 79 and just past N. Crosscut 415 W. Hard white quartz on foot-wall along side of drift	14	$6.40	$1.10	$7.50	105	16.327 ~~$163.27~~ for 60 inches
101H	Ferruginous ore in back of drive, contains small amount manganese	46	17.28	1.72	19.00	874	
		60				979 / 60 = $163.27 16.327	
262	10 East of 261. Soft oxidized ore, containing considerable iron oxide, and a little clay mixed with gossany quartz on footwall side of drive. Foot wall not reached	30	$5.05	$0.45	$5.50	165	$11.03
262F1	Horse of mineralized country rock, containing stringers of quartz	22	1.62	0.25	1.87	41	for
262F2	Hard quartzy ore showing iron stains and small amount of manganese. To hanging wall	32	20.00	2.50	22.50	720	84 inches
		84				926 / 84 = $ 11.03	
725	At N. Crosscut 115E samples across back of drift. Hard quartzy ore on hanging containing manganese streaks and some iron	30	$18.75	$1.75	$20.50	615	$12.16
725H1	Highly altered country containing numerous stringers of quartz and limonite	20	3.88	0.37	4.25	85	for
725H2	Soft ore, manganese banded with gossany quartz and clayey material	22	7.50	0.50	8.00	176	72 inches
		72				876 / 72 = $ 12.16	
725H3	Barren looking quartz on foot wall	20	1.13	....	1.13	23	To be rejected
		92					

Column 1 is the number of the sample. Column 2, description of the sample, copied verbatim from the underground note book. Column 3, width of sample in inches, feet, meters, or other unit. Column 4, gold value in dollars, shillings, or other unit. Column 5, silver value in the same unit of value. Column 6, total of both. Column 7, the unit of value multiplied by the width as the inch-dollar or inch-shilling. Column 8, average value and width for the interval.

In case the ore contains base metal of value, appropriate columns should be provided for noting percentages and 'inch by per cent' (width by per cent). Sufficient space should be left between the successive samples to note the results of any re-samples or re-assays and the results of such noted in red ink.

The assay results once transferred to this book from the assay office record permits of a quick determination of erratic distribution of valuable minerals. If any occur, a re-assay should be made of the particular sample in question, or it may be deemed advisable immediately to make a re-sample of the exposure. Those intervals that are re-sampled should be marked with the same number as the original and are differentiated from it by adding the letter *R* to the number; thus, if the original number is 147, the re-sample is marked *147R, 147 RF1, 147 RF2,* etc.

The calculation of the assay value of each individual *interval* is done by combining the results of the fractional samples, bearing in mind subsequent mining operations. If a high-grade streak occurs on one of the walls, it may be necessary, owing to the low assay value of the remaining portion of the orebody, to consider the advisability of stoping the high grade by itself. One cannot arbitrarily decide on such a procedure; the geological and mining conditions must be kept in view, as it may not be economically possible to mine one portion of the orebody alone.

These questions must be definitely settled at the time of calculating the interval assay values and widths, because they are used for the assay plan and for calculating the average assay value and average width of the whole orebody, and thereby enabling a determination of the total metal contents of the ore to be mined.

In averaging up fractional samples, profitable ore may show on either wall, with unprofitable ore between. If the low-grade streak be sufficiently wide, it may be possible and economical to allow it to stand as a pillar. If, however, the whole width of the orebody must be broken in the subsequent mining operations, the low-grade ore must be averaged with the high, in order to ascertain the breaking value of the ore for the subsequent calculations.

Values may tail off towards one wall, but with no visible geological change in the formation; in such case, the engineer must use his knowledge of the orebody and his mining experience to determine whether it will be possible to break the ground in such a manner as to leave the unprofitable ore standing. In arriving at a conclusion he may have in mind subsequent hand sorting to raise the grade of the ore after it is broken, but before it is sent to the mill; or he may deem it possible to break ore to a required grade guided by careful stope sampling. In either instance it should be remembered that a greater allowance for decrease in grade must be made than is shown by taking the proportionate widths.

Whether sorting be adopted, or careful stoping methods, a considerable admixture of the low-grade material is inevitable and the assay value will be diminished accordingly. Under such conditions the ore cannot be mined as clean as it can be sampled, and it can readily be appreciated that no sorting operation will enable the whole of the poor rock to be picked out.

Where the orebody is narrow and some of the wall rock must be broken to get a stope wide enough to work in, care should be taken to make ample allowance for the admixture of wall rock. In the narrow reefs on the Rand, it is customary to calculate the average assay value for a stoping width of 36 in., but actual practice shows that only about 75% of the calculated grade is obtained.* This is probably due to the fact that unless one is mining to a slip or head, no accuracy of dimension can be attained in ordinary mining work.

Diagram 1 illustrates the method of arriving at the average assay value based on actual practice and showing three different conditions. The figures are hypothetical. The calculation of the average of two or more samples is based on the premise that a mixture of materials of different value will have an average value directly proportional to the quantity and value of each of the components.

To illustrate this by a simple and concrete case, let us assume:

	1 ton	of	ore	assaying	2	oz. gold
mixed with	2 tons	"	"	"	1	" "
the average is not						
(a)	3 tons	"	"	"	1½	" "
but is						
(b)	3	"	"	"	1⅓	" "

(a) corresponds to the arithmetic average, while (b) represents the volumetric average which gives the correct result. The arithmetic average is arrived at by taking the sum of the assays and dividing by the number of samples. That this is incorrect can be seen by calculating the total contents as follows:

	Tons		*Assay*	*Total contents*
	1	×	2 oz.	2 oz.
	2	×	1 oz.	2 oz.
Totals	3 tons			4 oz.

If 3 tons of the mixture contains a total of 4 oz., each ton will have an average value of:

$$4 \div 3 = 1\tfrac{1}{3} \text{ oz.}$$

It is obvious that the richer material adds a greater quantity of gold to the mixture than the poor stuff and the arithmetic average takes no account of this.

Changing the above proportions, but keeping the same figures:

$$\begin{array}{lllll} & 2 \text{ tons of } 2 \text{ oz. ore} & = & 4 \text{ oz.} \\ + & 1 \ \text{"} \quad \text{"} \ \ 1 \text{ oz. "} & = & 1 \ \text{"} \\ \hline & & & 5 \text{ oz.} \end{array}$$

giving an average of $1\,{}^2/_3$ oz. per ton.

The averaging of mine samples is governed by the same principle, but instead of having the actual quantities to deal with, they are repre-

*E. J. Way, Trans. Inst. Min. and Metallurgy.

sented by samples which are weighted proportionally to the tonnage of ore they are supposed to represent, which in turn is directly proportional to the width sampled.

In the illustration just given, in each component the quantity was multiplied by the assay which gave a factor in this case the total gold which added together with the other factor arrived at in the same way and divided by the total quantity gave the average assay value.

In averaging mine samples, as the tonnage is directly proportional to the width sampled, the width is used instead of tonnage.

The mathematics governing this method is fully discussed both in H. C. Hoover's[1] and T. A. Rickard's[2] books and no good purpose is to be served by repeating it here. That the arithmetrical average gives an incorrect result has been demonstrated by the two problems already given.

To get the volumetric average, the following is the general method employed.

The assay value for each metal is multiplied by the sample width in inches. This gives an arbitrary factor called the inch-shilling, inch-dollar, inch-pennyweight, or inch-per-cent, as the case may be. The sum of these for the samples to be averaged, divided by the sum of the widths sampled, gives the average assay value of the lot. Americans generally measure widths in feet and decimals, in which case the foot-dollar, or foot-per cent is used as a factor.

Referring to Diagram *1*, sample No. 101 is a simple case where the two fractions will be mined together. To get the average, multiply 14 (width of 101) by $7.50 (total gold and silver value) which gives 105 inch-dollars, to which must be added 46 (width of 101H) multiplied by $19.00 (total gold and silver value of same) or 874 inch-dollars, giving a total of 979 inch-dollars. Divide this by 60, the total width (the sum of 14+46), and there results a quotient of 163.27, which is the average value of the whole interval for the total width of 60 inches.

Looking at the results of the interval 262, it is to be observed that the foot and hanging wall portion are pay ore while the central fraction 262F is below grade. As the whole of this must be mined together, the low-grade portion must be averaged with the other two. The total inch-dollars 926 divided by the sum of the widths 84 gives an average assay of $11.03 for the interval over a width of 84 inches.

Taking Interval 725 next, it is seen that on the footwall (sample 725 H3) there is a low-grade band of hard material lying adjacent to soft ore (samples 725, 725H, 725H2) and that in stoping operations this band of hard ore can be left standing. Therefore, as it will not be mined it need not be included in calculating the average value of the interval. The total inch-dollars for the three fractions averaged divided by the sum of the widths, 72 in., gives an average of $12.16 for a total width of 72 in. Of course, to obtain this result the mine management would have to work the mine accordingly.

On the other hand, it is obvious, were this material of the same physical character as the overlying fraction 725 F2, it would have to

[1] 'Principles of Mining.'
[2] 'Sampling and Estimation of Ore.'

be mined and, therefore, averaged in the result. Likewise, if there were a band of soft low-grade ore on the hanging wall side of the orebody it could not be depended on to stand in stoping and would have to be averaged in. These are typical problems, but each case must be decided on its merits by the engineer when correlating his results.

Check-Sampling.—Before proceeding with the calculations to determine general averages*, it is always advisable to check-sample, say, 10% of all the samples taken. This work should be distributed over various parts of the mine, and the re-sample should, if possible, be taken by a different man than the one who cut the original sample. If the assays of the re-sample check the original results fairly closely, say within 10%, it may be assumed that the sampling and assaying has been correctly done and the averaging up of assays in the different blocks of ground may be proceeded with.

Erratic High Assays.—The record book now shows the averages of the fractional samples for each different interval. These must be again scanned for erratic high assays. By this is meant the occurrence of a high assay amid comparatively lower ones. For instance, if the general run of assays along a portion of a particular level is in the neighborhood of $10 and among them is found one or two showing $25, the occurrence is erratic and the intervals should be re-sampled. This occurrence may be due to an unobserved high-grade bunch or to the accidental inclusion of an undue proportion of a known high-grade streak or of native gold in the sample.

It has been pointed out that the object of mine sampling is to determine the stoping value of the ore, so that if in a group of ten successive samples, eight showed a $10 assay and two a $25 assay, the two high assays would raise the general average from $10 to $13, equal to $3 per ton, or 30%. It is obvious to one familiar with the geology of ore deposits that the amount of ore corresponding to the portion showing this high value is insufficient when mixed with the remainder to raise the general average value of the whole quantity by that amount. As a general rule, such isolated or erratic occurrences of high-grade material may be expected to extend upward only for about the same distance as exposed in the working, but whatever the reason may be, it has been found by experience that the only safe method of procedure is to cut down the value of these high-grade samples to the general average of the others, that is, to $10 in the particular instance given.

In a paper entitled 'Some Sampling Results' by the late E. H. Garthwaite, appearing in Vol. XVI of the *Transactions* of the Institution of Mining and Metallurgy, some startling sampling results are shown, because the erratic high assays were not eliminated in the calculation of the general averages. Four examples are given, and in each one, the samples were taken at 2-ft. intervals and all the permutations for 2, 4, 6, 8 and 10-ft. intervals worked out, that is, 2 for 4 ft., 3 for 6 ft., 4 for 8 ft. and 5 for 10 ft. Mr. Garthwaite's

*The method of calculating the average value of a block of ground is discussed later.

paper is well worthy of study and illustrates in an illuminating manner the serious error that may arise from not following the method suggested above. There are interesting details in all of the cases which he cites, but for the purpose of this discussion it will be sufficient to point out the error in the manner of deducing the average result in his example No. 4.

We find that 62%, or 35 samples, of the total number taken, assayed less than 5 dwt. of gold per ton, while 91½%, or 52 samples, assayed less than 20 dwt. of gold per ton. Of the balance of five samples, representing 8½% of the total

1	assayed between	20	and	30	dwt.
1	"	40	"	50	"
1	"	200	"	300	"
2	"	300	"	400	"

Mr. Garthwaite states that his usual practice was "to reduce any high assays, say of 100 dwt., or over, to the average of two samples above and two below and (including the original) take the average of the five samples."

Averaging an erratic high result with two samples above and two below, does not meet the requirements of good practice. For reasons stated in the paper, which are not altogether good, he omitted doing even this. The exact assays of the five samples quoted are not stated in Mr. Garthwaite's paper; hence it is impossible to calculate the actual effect that their inclusion had on the remaining 52. But it may be laid down as a rule to be followed almost invariably in cases of this sort, that *every one of these five samples should have been eliminated,* or, in other words, reduced to the general average of the other 52. A study of the complete data, which is not available, may even reveal that some of the remaining sample values should have been reduced as well.

The paper shows the variations in the averages for the different permutations of the 2-ft. intervals, but those obtained in the different 10-ft. intervals will sufficiently illustrate the point at issue. They are as follows:

Average value dwt.	*Average width inches*
6.96	53.5
2.01	51.0
38.38	48.0
3.70	50.0
32.52	51.0

The average of the 2-ft. intervals gave—
17.47 dwt. over 49.0 inches.

As compared to these results of Mr. Garthwaite samples at 10 ft. intervals, taken by the mine staff, gave 6.83 dwt. over 51.5 inches and an independent sampling gave 5.41 dwt. over 53 inches.

To show the effect that the inclusion of these five high assays had on the general average of the lot let us work out their volumetric average, taking the lower figures of the limits given and assuming equal widths, as the exact widths are not available.

Thus:	Sample		Assay		Quantity × Value
	1	×	20 dwt.	=	20
	1	×	40 "	=	40
	1	×	200 "	=	200
	2	×	300 "	=	600
Totals	5				860

This figure 860 (being the sum of the quantity × value factors) when divided by 57 (the total number of samples taken) gives a quotient of 15—the number of pennyweight by which the average assay has been increased on account of the inclusion of the five samples. Of course, this is not quite accurate as no account has been taken of exact widths, but it does serve very plainly to indicate the large error that resulted from the inclusion of these erratic assays. Of all the samples 62% averaged less than 5 dwt. and, in order to increase the general average from 5 to 17½ dwt. (Garthwaite's average for 2-ft. intervals), equal to 250 %, the remainder of the samples must represent a much larger tonnage of high-grade ore than obviously they did.

In the paper there is no information which will enable one to account for these erratic high assays. They may have been due to the inclusion of an excessive amount of an easily sampled high-grade streak in the orebody or to the presence of coarse gold. As the independent sampling and the mine sampling show materially lower results, we must assume that in these last two samplings the high grade could not have played such an important part, either the sampling itself was more carefully done or the results interpreted more conservatively.

That the lower average value obtained by them corresponds more nearly to the breaking value of the ore, there can be little doubt, although sampling at 2-ft. intervals ought to give more accurate results than at 10-ft. intervals. Perhaps the moral to be drawn from this is, that excessive refinement in work of this kind is not always worth the effort, especially when omitting to use correct methods in part of the operation.

There are exceptions to every rule, and in some orebodies there is a persistent recurrence of erratic high assays, which, if excluded from the average, would erroneously show unprofitable ore. A case of this kind cited by Walter McDermott in the discussion of Garthwaite's paper is that of the Tomboy mine in Colorado. This property has been a consistent dividend payer for a number of years. The general mass of the orebody is of low grade, but there is a persistent recurrence of small bunches of high-grade ore, which of course show erratic results in sampling, but these must be averaged in with the low-grade samples to find the correct stoping value of the ore.

In discussing Garthwaite's paper, H. S. Munroe calls attention* to the formula for determining the probable error of the arithmetical mean of a number of observations, and states: "The use of this formula for the study of sampling data will give valuable information as to the accuracy of the final result and enable one to deal intelligently with high assays. If $n =$ number of observations, $d =$ difference of each from the arithmetical mean, then the

$$\text{Probable error} = 0.6745 \sqrt{\frac{d^2_1 + d^2_2 + \ldots d^2_n}{n\ (n-1)}}$$

In Garthwaite's second example, Mr. Munroe makes certain calculations showing the probable error by omitting one or more of the high assays, but the deductions he is able to make do not clear up the difficulty.

No doubt the formula will perform in a satisfactory manner the service for which it is intended, but applying it to find the probable error in a number of sampling observations, will not necessarily give a correct result. This formula to be of use necessitates as an hypothesis that the individual observations are in themselves correct, although it may seem to be a paradox, attempting to find the probable error in a number of observations, each one of which is assumed to be correct. With the sampling results with which we are here dealing, there can be no question that an error in sampling has been made, as conclusively demonstrated by the entirely different results that were obtained by the mine management and the independnt sampling over the same stretches of ground. Consequently any calculations based on the result of the incorrect sampling must likewise be incorrect. This is but another illustration of what has been pointed out so often in these pages, namely, that judgment must be used in sampling and valuation work. No formula can be a substitute for experience, and the mine management more familiar with the ore got the more correct result.

Assay Plans.—When the re-sampling has been completed and the check assays noted, the preparation of an assay plan may be undertaken, for not until then are the data for doing this complete.

When an orebody is vertical or dips at a high angle, it will be found generally more convenient to show the assays on a longitudinal projection or section rather than on a plan, because in the plan of steeply inclined workings, the levels overlap, or there is so little space between them that the necessary figures cannot be written in, or at best are so close together as to be confusing. Where the assays are shown on a projection or section, unless the levels are practically straight and also parallel to the plane of the section, they are shown foreshortened, so that the sampling intervals cannot be platted to scale but must be foreshortened as well. To facilitate this, they may first be laid off on a strip of cross-section or other paper and then transferred to the drawing.

*_Mining and Scientific Press_, Vol. 105; page 18 (July 6, 1912).

Some orebodies are so tortuous that there are curved lines not only along a level, but also in the winzes and raises, on account of changes in dip at different points along the lode. In such case not even a plan in an inclined plane can represent all the workings in their true length, and it may be necessary to show assays both on a projection and in plan.

With orebodies such as the Rand banket, or other more or less regularly dipping bodies, it is a common practice to plat the workings on the plane of the lode. With a highly inclined orebody of considerable width a separate plan of each level may be necessary. The engineer then must decide according to the conditions which is most suitable, namely to show the assays on a plan of the workings, on a projection, or on a longitudinal section.

The assays once platted furnish a graphic representation of the mode of occurrence of the values, and if the mine developments are sufficiently advanced, show the pitch of the ore chutes and other phenomena of a like character useful in a proper estimation of the tonnage of ore opened up. One level may show a persistence of values through its entire length, and the succeeding one may show a break, although the lode matter continues. This may be an indication that the orebody is pinching out as depth is attained. Likewise an intrusion may make its appearance on the lower level, which does not exist on the upper one. The orebody may pinch in places and the assay plan will show whether such phenomena are persistent from one level to the next, or are merely local. The assay values taken in conjunction with the geological features, which subject has been discussed more fully elsewhere, are the chief factors in a determination of the potentialities of the mine. They must be carefully studied before proceeding with the ore reserve calculations.

There is some divergence of method in setting out the information on an assay plan or section. My own practice is to omit the sample numbers and to note the assays and widths opposite the respective intervals they represent. The sample is given a number only for the purpose of identifying it during the various stages from the time it is taken till it is noted on the assay plan. The number itself has no excuse for existence on the assay plan, except for the possible contingency that one may wish to refer back to the original underground notes, but even in that case the sample can be quickly identified by its position with reference to the nearest survey station, cross-cut, winze or other convenient datum point.

The information wanted on an assay map is the assay value and width of the interval to compare it with its neighbors, and the picture conveys the full information desired. Where there are several metals the assays of each can be marked in a distinctively colored ink or the same sequence from left to right adopted throughout. I have seen the assays and widths enclosed in a circle and an arrow drawn to indicate the place from which they have been taken. Some engineers even go so far as to make an assay plan on which the sample intervals are marked with their respective sample numbers and the assays and widths set out in a tabulated statement. This method detracts from the value of the information, as it is difficult to follow.

The usefulness of an assay plan depends on the facility with which the information desired can be obtained. Because fractional samples have been taken is no reason why each individual result should be shown on the assay plan. Attention has already been drawn to the fact that fractional samples are taken, for reasons beyond the control of the engineer. From the mining viewpoint what is wanted is the average value of the orebody at the *interval*, and it is immaterial whether it has been necessary to take two or twenty fractional samples to get the value of the interval. Recently I had occasion to investigate some assay plans of a wide orebody. There were as many as 20 fractions 5 ft. wide to represent the whole interval, and before an intelligent idea of the values was obtained these had to be averaged. If there be more than one grade of ore, each of which is to be mined separately, then the outlines of each should be shown and the average of each interval set down accordingly.

Calculation of Tonnage.—Preliminary to calculating the tonnage of ore opened up, the mine is for convenience divided into blocks or sections, whose size is ordinarily determined by the physical features of the mine workings, the ground bounded by two levels and two winzes being usually a determining factor. The calculations are done in one of the 'Ore Reserve Calculations' books—and as an accessory those for individual lengths may be done on strips of cross-section paper, because the ruled squares permit laying off the intervals to scale. As the data for a given length of ground may be used twice, this permits bringing the proper strips together as required. For instance, the figures for a level lying between two blocks, one above and one below, are used for both blocks, the calculations for which are made at different times.

To get the *general average value* along a length of working, the same general method is employed as used in averaging the results of fractional samples, except that the two terminal samples are each given only half the weight of the others. The reason for this is, that a sample is supposed to represent the ore half way between it and the next sample; with the terminal samples ore only on one side is taken into the calculation.

To take a specific case, let us assume that it is desired to obtain the average assay value of a rectangular block of ground, bounded by two drifts 100 ft. apart and by two winzes in the orebody 200 ft. apart.

From the assay records the average value of each level and each winze is calculated, care being observed that the terminal samples of each lot are not weighted disproportionately to the tonnage they represent. The method of averaging the interval values to ascertain the general average of the section is the same as that used for averaging the fraction assays. The assay value of each interval is multiplied by the width, giving a width $\times$ value factor, such as inch $\times$ dollar or inch-shilling. The sum of these for all the samples is divided by the sums of the widths of the samples and the quotient is the average assay of the lot. The average width is found by dividing the sum of the width by the number of samples.

Where the vertical development openings are carried in the orebody in the same manner as the drives, then to obtain the average of the block of ground the winzes and drives are weighted relatively to their length. In the case assumed, as the two levels are each assumed to be of the same length and also the two winzes, the calculation may be simplified by averaging the two levels and the two winzes separately and then taking the general average as shown here—otherwise the method of averaging necessitates the multiplication of the length $\times$ width $\times$ assay for each level and each winze and obtaining the average assay by dividing the total of these factors by the sum of length $\times$ width in the same manner as shown below.

Take the average of the levels and winzes to be as follows:

Levels 200 ft. long average value \$10 for 5 ft.
Winzes 100 ft. high " " 8 " 4 "
Then $200 \times 5 \times 10 = 10{,}000$ sq ft.-dollars
$100 \times 4 \times 8 = 3{,}200$ " " "

13,200 sq. ft.-dollars.

To get the average value of the block of ground, divide this amount by the sums of the product of the length of each by the width in feet, thus:

$$
\begin{aligned}
200 \times 5 &= 1{,}000 \\
100 \times 4 &= 400 \\
\text{Total} \quad & 1{,}400 \text{ sq. ft.}
\end{aligned}
$$

therefore $13{,}200 \div 1{,}400 = \$9.43$ average value. To get the average width, divide the sum of the lengths by the sum of the products of each length by its corresponding width, thus:

$$
\begin{aligned}
200 \times 5 &= 1{,}000 \\
100 \times 4 &= 400 \\
\text{Totals} \quad 300 \quad & 1{,}400 \text{ sq. ft.}
\end{aligned}
$$

Dividing the sums of the lengths into the product we have:

$1{,}400 \div 300 = 4.67$ ft. average width

Therefore this block of ground 200 ft. long by 100 ft. high averages 4.67 ft. in width for an average assay value of \$9.43.

The cubical contents of the block of ground is obtained by multiplying together the three dimensions, namely the length by the height of the average width. This gives the number of cubic feet of ore in the block, and the number of tons is found by dividing the cubic feet by a previously determined factor, representing the number of cubic feet to the ton.

With quartz 13 to 15 cubic feet per ton of ore in place is ordinarily used, 14 being a common figure. The proper factor must be determined in each particular case, and if no mine records are available, then specific gravity determinations must be made by the engineer and due allowance made for the cracks, vugs, and other open spaces in the orebody.*

In the case under discussion, let us assume the ore to be quartz and 14 cu. ft. of ore in place, to weigh one ton of 2,000 lb.

Then 200 ft. long × 100 ft. high × 4.67 ft. wide = 93,400 cu. ft., and 93,400 ÷ 14 = 6,671 tons of ore, having an average assay value of $9.43.

Needless to say, underground workings are not always laid out so symmetrically; very often one level in a block is longer than the other, but the same procedure is followed, each being weighted proportionally to the length of face taken into the calculation.

The volumetric method of averaging assays, as described above, is the one in general use among engineers. H. S. Munroe,[1] in a paper read before the Mining and Metallurgical Society of America, states that when it is required to average the assays of different kinds of ore varying considerably in specific gravity this method may give incorrect results, as under such circumstances it may be necessary to take the specific gravity into the calculations as a factor. The publication of this paper serves to bring forward a most important theory and one of which engineers in general have as yet made but little use. Since the date of its publication, I have had the opportunity of discussing this question with its author at considerable length by exchange of letters, and in this correspondence Mr. Munroe amplifies the theory and gives the method a much wider application than is to be inferred from the original article. In fact, he now proposes to apply the gravimetric method in the calculation of averages, wherever there is any difference in the specific gravity of the constituent minerals in the ore.

The theory underlying his method is contained in the following quotation from one of his letters:[2]

"If we have a vein with ore of uniform density, varying in *width* from 2.6 to 4.0 ft., there is no question that we should weight the samples according to the width of the vein, as the wide places represent more tonnage than the narrow. If we have a vein uniform in width but varying from 2.6 to 4.0 in *specific gravity*, it is equally true that the samples of high specific gravity represent a higher tonnage of ore."

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*See chapter on specific gravity determinations (page 137).
[1]*Mining and Scientific Press,* Vol. 105; p. 18 (July 6, 1912).
[2]Letter dated April 29, 1913.

On the face of it, this is a most convincing statement and would seem to leave but little room for argument.

Before proceeding with the discussion of this theory, it will be useful to quote one of the cases cited by Mr. Munroe in his paper, to show the application of the method. He states:

"For the purpose of illustration, I assume five samples taken at uniform distances in a galena vein, varying in thickness from 10 to 60 in., and the ore varying from 10 to 80% of lead.

	Thickness, Inches.		Lead, Per cent.		Inch. Per cent.
	60	×	10	=	600
	20	×	40	=	800
	50	×	25	=	1250
	10	×	80	=	800
	10	×	60	=	600
	5/150		5/215		150/4050
Averages	30		43.0 (Arithmetic)		27.0 (Volumetric)

"Calculating the average in the usual way by multiplying the thickness in inches by the percentage of lead and dividing by the total number of inches, an average value of 27% is found, which may be called the volumetric average. This is less than the arithmetic average, which in this case is 43%. Neither of these results, however, is correct, as the result obtained by sampling must be given weight in proportion to the tons of ore which each sample represents. In the following table I have introduced specific gravity as well as thickness:

Thickness, Inches.		Specific Gravity.		Inch-Gravity.		Lead, Per cent.		Inch-Gravity Per cent.
60	×	3.0	=	180	×	10	=	1800
20	×	5.0	=	100	×	40	=	4000
50	×	4.0	=	200	×	25	=	5000
10	×	7.5	=	75	×	80	=	6000
10	×	6.0	=	60	×	60	=	3600
150				150/615				615/20400
		Average spec. grav. 4.1				Average per cent 33.17		

"If the thickness in inches is multiplied by the specific gravity of the sample, a product is obtained which may be called the inch-gravity value, which is a unit proportional to the tonnage represented by the sample. Multiplying this by the percentage of lead found by assay gives an inch-gravity-per-cent product. Dividing the sum of these last products by the total of the inch-gravity column gives as the average 33.17%, which lies between the volumetric and the arithmetic average. This may be called the *gravimetric* average."

In the averaging of assays, we are really confronted with two problems, first the averaging of fractional samples and the other the averaging of interval samples. The first problem is really doing by mathematics what we actually accomplish in the operation of cutting the sample, that is, we endeavor to get such quantities from each unit of width of the ore as will be proportional to the actual tonnages that will be mined later on. In cutting the sample, as has already been explained in a previous chapter, these quantities are taken by volume; therefore, those constituent minerals which have the highest specific gravity will comply with the condition laid down in Mr. Munroe's theory and influence the result to the greatest extent; in other words, an equal volume of heavy mineral will add a greater weight to the sample than a light mineral.

What is meant to be conveyed by the above is that in any ore-body where there is a mixture of minerals, if the sampling is properly done, representative amounts must be taken and only in case the ore is banded in such a manner that separate samples may be taken of the different bands, can the specific gravity factor seriously influence the result.

In averaging interval samples a somewhat different condition exists. Mr. Munroe shows, although citing an extreme case, how much the element of gravity may affect the result. On the other hand, in his correspondence he advances arguments in favor of its use in every-day work with any kind of ore where there is any mixture of minerals. He states:*

"My plan of operations in the case of a concentrating ore, for example, in which the valuable mineral is scattered through the vein filling, is to cut a groove with the moil across the vein from foot to hanging wall. This we will consider one sample. Five or ten feet away, as the case may be, I cut another sample in a similar manner, and so on. In each sample, I determine the average specific gravity of mineral and gangue together (not separately), using for this purpose an average sample from one of the rejected portions of the moil sample when working the whole down to bottle sample for analysis. The sample upon which the specific gravity determination is made thus represents the average specific gravity of the moil sample from one wall to the other. The same thing will be done with each moil sample, so that we shall have as many determinations of specific gravity as samples are taken in the mine, several hundreds or several thousands perhaps. In averaging the assay values I do not depend on the measurement of the vein from foot to hanging wall alone, but multiply this measurement in each case by the specific gravity of the corresponding sample, obtaining thereby an inch-gravity factor which is proportional to the *tonnage* and not to the *volume* which each sample represents. In other words, my final average is a gravimetric one and not a volumetric one, as is usually the case.

*Letter dated May 16, 1913.

"It is not true that the difference between the volumetric and gravimetric methods will be negligible even with a small percentage of valuable mineral present. The common gangue minerals, silicates, carbonates, etc., of the vein filling differ very greatly in specific gravity. You have only to refer to Dana to establish this fact. Different samples of ore from the same level will vary in gravity with the proportion of one or another gangue mineral present. Again the vein filling often consists largely (sometimes wholly) of barren sulphides or sulphides containing small percentage of valuable mineral only. The presence of a greater or less quantity of these low-grade sulphides (which in such case are as much gangue minerals as the silicates which may be present at the same times), does affect very materially the specific gravity of the ore; and adjoining moil samples with equal vein width represent great differences in tonnage—as great as those due to width which we are accustomed to take into consideration. For example, the concentration ratio at Great Falls and Anaconda is about 3 to 1. The concentrate is almost wholly mixed with metallic sulphides and the tailings silicious material. The concentrate does not contain more than 10% of copper, so that the metallic sulphides in the ore and in the concentrate are largely gangue. Any concentrating ore which contains so large a percentage of metallic sulphides, must vary greatly in specific gravity all through the mine, the variation perhaps running from that of the silicates with little or no sulphides to clean sulphides with little or no silicate. The percentage of the valuable mineral, in this case copper, will vary but little, say one per cent to five per cent. Such a mine would give as extreme a case as any that I have indicated in my original paper. But we do not have to consider such extreme cases. I venture to say that the ore of every mine, whether containing sulphides or not, will be found to show wide variations in specific gravity, if specific gravity determinations be made on every sample, as I propose. If I am right in this, my method will have universal application wherever we desire to introduce all possible precautions to secure the most accurate results practicable. This is my attitude at all times, especially when it adds little to the work and little to the expense, as in this case."

That the use of this method will require an additional amount of expense and time is no valid reason against its application, provided it gives a more accurate result than any other. Taking the case of the galena orebody just quoted as an illustration, it will be seen that the gravimetric average gives a considerably higher result than the volumetric, and in any similar cases, similar results must be obtained, of course, varying only in degree. No doubt conditions might be assumed where the gravimetric average would show a lower result than the volumetric, but certainly by far the largest number of cases would show a higher result.

It is a well known fact, which has been fully explained in the preceding pages, that the assay of ore as mined is usually lower than shown by the sampling of the ore before it is broken. It will be recalled that in the case of the Rand this difference amounts to as much as 25%. If engineers were in a position to make a proper allowance, it is safe to say that it would be done. Every engineer in assessing the assay value of an ore deposit, always makes an allowance for the admixture of country rock with the broken ore and yet this difference continues to exist, so much so that most engineers will state that an orebody cannot be mined as clean as it is customary to sample it. Of course, the object of mine sampling must be to determine the stoping value of the orebody, and proper scientific methods aim at making these sampling results correspond with the stoping results.

The error being all on one side, leads one to the conclusion that the difference is largely beyond the control of the engineer and that despite any precautions that may be taken, an excess of the brittle mineral, which is usually the richest, enters the sample. This brittle mineral is, of course, usually a sulphide with a higher specific gravity than the ordinary silicious or other non-metallic gangue and this may account for the sampling results being higher than the stoping results.

The application of the gravimetric method in the majority of the cases will increase this condition and necessitate the application of a larger factor of reduction.

There is no use in altering one's methods unless they show an improvement over the ones that have previously been in use. While I am frank to admit that the argument in favor of the gravimetric method is an extremely strong one, yet considering the greater amount of work involved when using it and the fact that apparently it does not bring us nearer to the stoping values, makes me hesitate in recommending its use. On the other hand, I feel that the method has not been sufficiently tested to demonstrate its usefulness, or otherwise, and until greater experience is had with it, judgment must be suspended.

Inaccessible Ore.—We now come to the discussion of certain problems, arising from the interpretation of the assay results. It may eventuate that portions of the workings have not been sampled because inaccessible on account of caved ground, timber, or other cause. The question arises as to how these shall be taken into the ore reserve calculations. It may be arbitrarily stated that where no samples are taken, no value should be allowed, but this is often manifestly unjust. Of course, if good grade ore is reported to exist behind timber, while on either side low-grade ore or barren ground is exposed, such statement must be looked upon with suspicion and the weight of evidence would lead the engineer to omit this stretch of ground from his tonnage calculations.

On the other hand, where there is a stretch of timbered ground, on both sides of which is ore of minable grade, the engineer may perhaps be justified in assigning to this portion the average value of the neighboring ore, provided that it is not too long. It must be borne in mind that the working may have been timbered purposely to obscure an unprofitable stretch of ground. Here again judgment is called for, and if there is any doubt, the tonnage corresponding to this length should be reduced by a definite percentage according to circumstances or preferably omitted altogether. It will be noted that a reduction in the tonnage is suggested and not the value, although that may be deemed advisable as well.

At times the engineer has the result of a previous examination or the mine records to guide him. As an illustration of the difficulty that besets one and the risk in making assumptions of this character, I recall a case where the result of a previous examination showed lower assay values in ground that was timbered at the time of the second engineer's visit than the average of the neighboring ore. In such case the lower values of the former examination may be used, providing the work was done by a reliable engineer.

If the procedure suggested above is followed and the margin of profit in the ore is not too small, any discrepancy arising therefrom is not likely seriously to affect the profit calculated for the block, because the deduction made as a factor of safety simply gives a smaller tonnage of higher grade ore. If a sufficiently conservative deduction has been made the net profit shown is not likely to be very different except in the single instance where the margin of working profit per ton of ore is very small. In such an event this feature must be given due weight in the engineer's calculations and he must not hesitate if circumstances warrant in omitting such a section altogether.

It is in cases of this sort that the skill of the engineer is brought into play. He must be guided by the geological evidence as well as the mode of occurrence of values, to enable him to form a proper opinion and in the absence of positive evidence as to values he must err, if at all, on the side of safety. Better be sure than sorry!

Irregularly-Spaced Drill Holes.—In churn-drilling the disseminated copper deposits which form the basis of the operations described in Chapter IX, expense is not spared in preparing the ground, building roads, etc., in order to space the holes at regular intervals. Sometimes drilling tabular deposits cannot be carried out in such a systematic manner, owing to physical conditions, or other causes, in which case the method of averaging up the assays must be modified to meet such irregularity.

To average the assays of holes spaced at regular intervals, the volumetric average of the samples is taken in the same manner as with samples at regular intervals in ordinary underground workings; that is, the interval or distance between the samples is not taken in as a mathematical factor, because it influences each sample in a like proportion. On the other hand, with irregularly spaced drill holes, the distance between the holes must be allowed for, as otherwise important differences may result.

The theory underlying the sampling interval and its influence on the general result has already been discussed elsewhere. Each sample is supposed to represent the block of ground half way to the next sample, and the same rule must apply in considering the results from drilling operations. In the case of irregularly spaced holes, a trapezoid must be considered, whose dimensions are the mean thickness of ore at the two holes and the distance between them. The samples, of course, must be weighted in the calculations in direct proportion to the quantity of ore they represent; hence the area of the trapezoid multiplied by the volumetric average assay of the two holes gives what may be called the area × per cent factor.

The following example will indicate how easily miscalculations may occur if the results of the drilling are not properly interpreted, and the assays weighted proportionally to the averages they represent.

Assume four holes put down in a copper deposit by churn drill, or diamond drill, and spaced as per sketch.

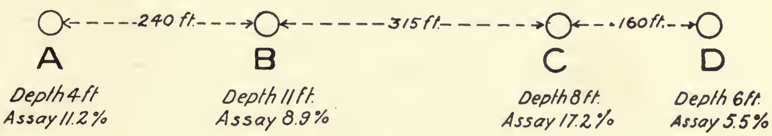

Fig. 21.

Thus we have average depth and assay of adjoining holes.

Depth, Ft.	Assay per cent.	Foot × per cent.
4 ×	11.2 =	44.8
11 ×	8.9 =	97.9
2/15		15/142.7
7.5 ft. av. dpth.		Av. 9.5%

Depth, Ft.	Assay per cent.	Foot × per cent.
11 ×	8.9 =	97.9
8 ×	17.2 =	137.6
19		19/235.5
9.5 ft. av.		Av. 12.4%

Depth, Ft.	Assay per cent.	Foot × per cent.
8 ×	17.2 =	137.6
6 ×	5.5 =	33.0
14		14/170.6
7 ft. av.		Av. 12.2%

To get the average of the four holes, allowing only for the ground within them, there follows:

Aver. Depth Ft.		Distance Between Holes.		Area of Trapezoid.		Average Assay.		Area × Per cent.	
7.5	×	240	=	1800.0	×	9.5	=	17100	
9.5	×	315	=	2992.5	×	12.4	=	27107	
7.0	×	160	=	1120.0	×	12.2	=	13664	
24.		715		/5912.5		5912.5		/57871/9.8%	Real Average

8.3 ft. average depth.

As compared to this result, by omitting the distance between the holes, we have the following:

Hole.	*Depth Feet.*		*Assay at Hole.*		
A.	4	×	11.2	=	44.8
B.	11	×	8.9	=	97.9
C.	8	×	17.2	=	137.6
D.	6	×	5.5	=	33.0
	4/29				29/313.3/10.8% *average.*
	7.3				

This shows a difference of 1 unit of copper, or 10%, approximately, of the total.

If we only weight the two end holes one-half we get:

A.	2	×	11.2	=	22.4
B.	11	×	8.9	=	97.9
C.	8	×	17.2	=	137.6
D.	3	×	5.5	=	16.5
	24				24/274.4
					11.4% *average.*

making a difference of 1.6 units of copper.

Vary the assays and the dimensions, and different results follow. Whether the two end holes shall be given only half weight must be decided in each individual case, depending on geological conditions. The differences do not always work out in the one direction, as the real average may be more than the incorrect one.

The volume of ore included within certain drill holes is determined by means of the prismoidal formula, commonly used for calculating cuts and fills in railroad work. The volume being known, the tonnage follows by dividing the amount by the factor representing the number of cubic feet per ton of ore in place.

CHAPTER XII.

ORE IN SIGHT.

Definitions.—If the estimation of ore merely consisted of calculating the contents and assay value of ground opened on four sides, the operation would be comparatively simple. The determination of what properly constitutes 'Ore in Sight' requires the utmost skill and experience on the part of the engineer, because a knowledge of the subsequent metallurgical treatment, as well as the economic conditions governing the operations, is essential. The monetary value of the 'Ore in Sight' governs the worth of the mine, hence its proper determination is a most important factor in mine valuation.

The problems that confront us are—how far beyond the existing mine openings can ore be assumed to continue, and can it be assumed to be continuous between the openings?

What is 'Ore in Sight'?

In 1902, as a sequel to the discussion of a paper presented by J. D. Kendall, the Institution of Mining and Metallurgy issued a circular to its members, recommending the following use of the term 'Ore in Sight':

1. That members of the Institution should not make use of the term 'Ore in Sight,' in their reports, without indicating, in the most explicit manner, the data upon which the estimate is based; and that it is most desirable that estimates should be illustrated by drawings.

2. That as the term 'Ore in Sight' is frequently used to indicate two separate factors in an estimate, namely:

(a) Ore blocked out, that is, ore exposed on at least three sides within reasonable distance of each other,

(b) Ore which may be reasonably assumed to exist though not actually blocked out,

these two factors should in all cases be kept distinct, as (a) is governed by fixed rules, while (b) is dependent upon individual judgment and local experience.

3. That in making use of the term 'Ore in Sight' an engineer should demonstrate that the ore so denominated is capable of being profitably extracted under the working conditions obtaining in the district.

4. That the members of the Institution be urged to protect the best interests of the profession by using their influence in every way possible to prevent and discourage the use of the term 'Ore in Sight' except as defined above; and the Council also strongly advise that no ambiguity or mystery in this connection should be

tolerated, as they (the Council) consider that such ambiguity is an indication of dishonesty or incompetence.

This was a step in the right direction and has served a most useful purpose, not only by making engineers more careful, but has been of educational value to the general public.

Other terms are used, but probably those most commonly in use must be credited to B. B. Lawrence, and form a contribution to T. A. Rickard's 'Sampling and Estimation of Ore in a Mine.' They are as follows:

1. Positive ore—Ore exposed on all sides.
2. Probable ore—Ore exposed on two or three sides.
3. Possible ore—Ore below the lowest level, or, as Mr. Argall expresses it, "below the last visible ore."

To my mind, neither of these sets of definitions is satisfactory. From a technical standpoint, the rules laid down by the Institution leave a great deal to be desired, and the reasons given why the two factors (*a*) and (*b*) should be kept distinct appear to me to be incorrect. The whole question of the estimation of 'Ore in Sight' must always be dependent most largely on the personal equation of the engineer making the estimate. We cannot apply *fixed rules* to serve as a guide; otherwise a novice could do the work, instead of it requiring great skill and experience. Local experience, as suggested by the Institution, is helpful, but in most cases the mining engineer has no local experience, and must depend on his general experience to guide him, notwithstanding which he is required to arrive at a definite result. Mr. Lawrence's definitions are altogether too restrictive and do not allow sufficiently for varying conditions. It is an attempt to arbitrarily define that which must depend on individual judgment.

Ore exposed on three or four sides cannot always be reckoned as positive ore or proved ore, for we can easily imagine the mine openings at such distances from each other that no conservative engineer would attempt to classify the included orebody as such. Geological conditions and character of mineralization must be taken into account. In the case of an iron mine, or a low-grade disseminated copper deposit, we may be willing to accept the results of holes drilled at intervals of 200 ft. as sufficient evidence to designate the ore as proved ore, and, on the other hand, we may be unwilling to so classify a body of gold-bearing quartz, of high grade, that was opened up with levels 150 ft. apart and intersected by winzes at intervals of 200 or 300 feet. In the impregnation type of deposit, such as those at Goldfield and Tonopah, Nevada, or of the Talisman mine at Karangahake in New Zealand, it certainly would be risky to follow this procedure with any such distance between openings.

The definitions laid down by H. C. Hoover in his 'Principles of Mining' seem to me to be much more satisfactory than those of the Institution of Mining and Metallurgy, because they are not restrictive and fully recognize that the assessment of ore tonnage is a matter of personal equation and cannot be bound by any hard and fast rules. Mr. Hoover's classification is as follows:

Proved ore: Ore where there is practically no risk of failure of continuity.

Probable ore: Ore where there is some risk, yet warrantable justification for assumption of continuity.

Prospective ore: Ore which cannot be included in the above classes, nor definitely known or stated in any terms of tonnage.

Proved ore instead of positive ore is more expressive; besides, the only positive ore in a mine is that which is already mined.

The expression 'probable ore' is one whose use has been sanctioned by long practice, but in view of the fact that the general custom in valuing mines, is to combine the tonnages and available profit of both proved and probable ore, to arrive at the total profit available and as a measure of the price which may be paid for the mine, it seems to me that in its present form of application it has a poor excuse for existence. If we treat probable ore like proved ore, then from the valuation standpoint they do not materially differ one from the other.

The expression 'possible ore' in itself seems to me to be particularly objectionable, because most people know that in mining all things are possible, and that the tendency is all on the side of the falsification of indications. On the other hand, 'prospective' ore is to be commended, because it again implies the necessity of judgment on the part of the individual and is a better mining expression.

No definite rules can be laid down to guide the inexperienced as to how far from the sample face toward the centre of a block of ground is permissible for ore to be called proved ore. It depends entirely on the character of the deposit. The two extreme cases already quoted will give some idea of the difficulty involved. Each man must be guided by his own experience. A not uncommon practice with engineers in ordinary base-metal deposits carrying precious metals, or medium-grade gold deposits, is to consider as proved, ore which is included between levels 100 ft. to 125 ft. apart.

The number of winzes necessary is largely a matter of geological conditions. It will readily be appreciated that some winzes are necessary, in order to prove that the ore does not occur in floors, or in no other way except as a continuous body. With a mine having levels 100 ft. to 125 ft. apart and winzes 200 ft. to 300 ft. apart, I think the usual practice will approve of calling all the ore included within such openings as 'proved' ore. (In special cases even these distances might be considered excessive.)

Probable ore is a guess on which a definite value is placed. When an engineer includes a definite tonnage of probable ore in his calculations, he is simply backing his opinion, that the values will continue sufficiently beyond the limits of the existing workings, to demonstrate that tonnage. It is customary with many engineers, where the headings are still in ore, to allow a certain distance beyond the faces for a further extension of the orebody. Included in 'probable' ore, too, is the extension of the orebody in depth, which also is a guess, based on empiricism.

More risk is involved in assuming the extension of the ore along the strike, beyond the faces of the drives, than in assuming its extension in depth. Experience teaches us that there are orebodies of all lengths and widths, yet there is no relation between the length and width of an orebody, that will guide us in making an estimate of this kind. On the other hand, as a result of the same experience, we have come to believe that usually in *fissure* veins there is a more or less definite relation between the length of the ore-shoot and its extension in depth. This put in figures would indicate a ratio of about 1 to 1 or $1^1/_3$, that is, if an orebody in a fissure vein is 1,000 ft. long, it may be expected to extend to a depth of 1,000 ft. or 1,300 ft. With types of deposit other than fissure veins, we have no guide except the experience of the engineer and his ability to size up the geological conditions. The cases of the Bellevue and Vivien mines in Western Australia have already been quoted.

In a formation like the Rand banket, one would be justified in assuming as probable ore an amount which would not be advisable in any other type of deposit, for here we have extensive developments, not only along the strike of the deposit but also workings in depth have proved a regularity in value which serves as a guide to the engineer in his estimates. At the great native copper mines of Lake Superior, despite the fact that the Calumet and Hecla lode has been worked for a number of miles on its strike and for a depth of more than two miles on its dip, with value throughout, the deep shafts of the Tamarack company sunk to depths of over 5,000 ft. vertically, in order to work the deeper portions of this deposit, have been unprofitable. The grade of the ore at that depth had decreased. These shafts were sunk for what the capable engineers in charge of the operations considered was probable ore, and enormous sums of money were expended in the attempt to realize on this probable ore, which the development proved to be unprofitable.

The problem with which the engineer has ordinarily to contend is simpler than this. From a study of the geological conditions and a consideration of the amount of ground that has been opened up, he must estimate how far the ore in the mine will extend beyond the existing openings. This will represent what he considers probable ore. Whether he will take into his calculations the entire tonnage estimated in this manner depends on the element of risk involved. A man may feel morally certain that the ore exposed on the bottom level of a mine is likely to continue to a considerably greater depth, yet may not feel justified, in view of the hazardous nature of any mining undertaking, in putting a monetary value on more than one-third or one-fourth on such tonnage.

One or two illustrations may serve to point the above discussion. In a replacement deposit, where the lowest workings were only about 250 ft. below the outcrop, and the examination demonstrated the existence of an ore-shoot 1,700 ft. long and averaging 5 ft. wide, the engineer valuing the property did not take into his ore reserve calculations an extension of more than 50 ft. below the bottom level. This he called probable ore. This, no doubt, was quite conservative, especially in view of the fact that the indications were that the zone of maximum enrichment had not yet been reached, and there was every likelihood of finding higher-grade ore at the next level. Among these indications may be noted that the bottom level of the mine showed better values than any above it. Of course, the question in all these forecasts as to what an orebody will do is, whether the same length, width and value of ore-shoot will be maintained.

A case of another kind is the estimation of probable ore in a mine like the Broken Hill South Blocks property, Broken Hill, New South Wales. The mine adjoining on the north is the Broken Hill South mine, which has been opened up to a depth of more than 1,000 ft. with its ore-shoots pitching towards the South Blocks ground. In view of the extensive nature of the underground workings on this famous lode and the evidence thereby afforded, it was perfectly justifiable in the case of the South Blocks mine, at the time of its purchase by an English company a few years ago, to assume a likelihood of a continuance of the ore for at least 300 ft. below the then bottom, or 400-ft. level. Development work since then has proved the correctness of this assumption.

When a block of ground is opened up on two sides as by a drift and a winze, it is usual to consider half the rectangle as proved ore and to take it into the tonnage calculations on that basis, namely, the quantity included in the triangle made by the development openings and the straight line joining their ends. The size and character of the orebody, and the extent of the development must to a considerable extent serve as a guide. In a mine where only a meagre amount of development work has been done at shallow depths from the surface, it is hazardous to make any great allowance for continuity of ore beyond the existing openings, because so little information is available. On the other hand, it may be equally risky to make any estimate in an old mine of considerable depth, despite the large amount of evidence available, because of the likelihood of the limits of the profitable ore having been reached.

It will be seen from the foregoing that even ore opened on only two sides may be considered proved ore. Probable ore, on the other hand, may be exposed only on one side or only at one point, that is to say, our most common application of the term is to ore below a level, or beyond any development opening, where the geological evidence affords reasonable proof for the belief that the ore will extend to the distance assumed. Prospective ore is anything beyond that. Only in rare cases can a mine be purchased at a price represented by the net profit contained in the proved and probable ore. There is a certain amount of risk, commonly known as mining risk, which generally must be assumed in the purchase of mining property. To offset this, there is the prospective ore, which, of course, is the likelihood of the ore continuing to a point beyond that considered safe to assess in the ore reserve calculations. The likelihood of an ore deposit continuing in depth depends on the geological conditions. To attempt to express this in definite quantities is, as Mr. Hoover states, an attempt to convey "an impression of tangibility to a nebulous hazard."*

What Mr. Hoover means undoubtedly is, that this prospective ore cannot be stated in the same definite terms of tonnage as are the proved and probable ore. However, we do give it a definite value in most cases by paying a portion of the purchase price for it, namely, the likelihood that the development will prove the existence of the additional amount of ore required to return the capital invested, plus a reasonable profit.

If we were dealing with a deposit, such as coal, we could assume the extension of the coal to a considerable distance beyond the existing openings, because we know that ordinarily it occurs with a regularity unknown in metalliferous deposits. In making such an assumption the risk is a comparatively small one and is principally restricted to geological disturbances, which, on account of the ease with which they can be determined beforehand, minimizes such risk. The Rand banket can be calculated with a considerable amount of certainty, although the development in the past few years has shown that practically all the former estimates as to the likelihood of the continuance of ore in depth have been wrong, because the assay of the ore has decreased greatly as depth has been attained, although the 'reefs' are as strong as ever. With the ordinary type of mineral deposit dealt with by the valuing engineer, the data available does not permit of a prognostication with any such degree of certainty as in the two cases here cited, therefore, the more skillful the engineer, the more accurate his determination of the prospective value of a mine. It is in this respect we find some of the greatest differences between the reports of different engineers on the same property. Aside from the natural conservatism of some men, which always leads them to make an estimation as safe as possible, a great deal is dependent on the skill of the engineer.

No rule can be laid down as to what properly constitutes prospective ore. This varies in each individual case, and, as reiterated, is dependent on the geological conditions. If there was any certainty of the prospective ore, it would, of course, be classed with either proved or probable ore, but in view of its uncertainty, it must be expressed in indefinite terms, certainly so far as regards tonnage. On the other hand, it is essential for the valuing engi-

*'Principles of Mining,' p. 18.

neer to express in a concrete form the idea of prospective value and this may be done by stating that, providing the orebody maintains the same dimensions and assay, as in the existing workings, then each 100 ft. of extension in depth will add a given tonnage to the available ore reserves. Of course, this may be expressed in units of 1 ft., but a unit of 100 ft. seems preferable, as more nearly approximating working conditions.

Where the zone of secondary enrichment has not been passed through, it is hazardous to make an assumption as to the value, but once the workings have penetrated into the unaltered ore, a greater degree of accuracy is likely to ensue from any forecasts that are made. In another chapter, the question of secondary enrichment has been discussed and must be borne in mind in connection with estimates of prospective ore.

Ore Reserve Plans.—From the assay plan the limits of the proved and probable ore are determined and the mine is divided into blocks, the size and shape of each being to a large extent governed by the mine workings, and each block is given an appropriate number for identification. The tonnage, average assay, and dimensions of each is calculated and noted on a special plan known as the ore reserve plan, or distinctive colors may be used to represent different ranges of values, so that the approximate grade of each block is seen at a glance.

Intermediate between the assay plan and the ore reserve plan are the calculations to determine the tonnage and average assay value of the ore. Generally speaking, ore cannot be mined as clean as it can be sampled, and for various reasons, fully explained in the preceding pages, the sampling results are usually higher than the breaking value of the ore. It is customary, therefore, to make certain deductions from the calculated averages as obtained from the assay plan and records, and this takes the form of an arbitrary reduction in grade, which may amount to from 5 to 25%.

Every engineer believes his samples to be correct, and the necessity for reducing the assay is ascribed to the fact that in mining, a certain amount of the country rock is bound to become mixed with the ore, because of the blasting and on account of the natural weakness of the walls. The amount of wall rock that mixes with the ore and dilutes it, depends on the geological conditions and the method of mining. It has already been pointed out that on the Rand the stoping value of the ore is sometimes only 75% of the sampling results and in other districts similar discrepancies are found. As a usual thing no account is taken of the increase in the tonnage on this account. For instance, if the ore reserves of a mine, calculated from the assay plan, amounted to 100,000 tons of an average value of $20, then if a deduction of, say, 10%, or $2, is made to give the stoping value, there results, according to general practice, 100,000 tons of $18 ore.

If there are actually 100,000 tons of $20 ore *in situ,* then this is incorrect as can be seen by calculating the total gold contents in the ore, thus, 100,000 tons at $20 contains $2,000,000. After dilution by the barren wall rock, there would still be $2,000,000 gold in the ore, but its average value as broken has been reduced to $18. If we divide $18 per ton into the total $2,000,000 in the reserves, there results 111,111 tons, so that it is assumed 11,111 tons of waste rock will get mixed with the ore, allowing the sampling results to have been correct, yet the extra tonnage is rarely taken into account.

Another method sometimes followed is to allow for the admixture of the 10% of waste rock. This gives 110,000 tons, containing $2,000,000 of gold, which by calculation shows an average assay value of $18.17 per ton. By this method advantage is taken of the whole amount of gold and the whole of the calculated tonnage, and may be justifiable, provided the ore has been sampled properly, and the samples not salted by the brittle mineral in the deposit.

It is a question in my mind whether the reason that the samples assay higher than the broken ore is not in most cases due as much to unconscious salting of the samples during the sampling operations as it is due to the admixture of wall rock. One rarely hears of a mine stoping higher than the sampling results and a great many errors are made at this stage of the work, as generally too little allowance is made and the ore does not stope to the grade assigned to it. With firm, smooth walls less deduction need be made than where the walls are bad, or the ore frozen to the walls, or where values shade off into the country.

As the calculations proceed, the various blocks of ore are tabulated, each under its appropriate heading, with the number of tons in each block and its average value. The total tonnage in the ore reserves is obtained by adding together the total of the proved and probable ore. The average value is determined by the same method as used in averaging assay widths and values described in Chap. XI (page 83), except that here we have tons instead of a unit of length. Thus there ensues a certain tonnage of proved and probable ore of a calculated average value from which the gross metallic contents can be determined, and once we are aware of the percentage of the metallic contents that can be recovered, and the cost of doing so, we have a measure of the value of the mine.

With base-metal mines, such as copper, lead, zinc, tin, etc., a basic price of the metal is sometimes assumed for purposes of calculation. No one can forecast with any degree of certainty the future course of prices of metals; hence any assumption as to price is a mere guess that may be wide of the mark, yet in order to convey a concrete impression of monetary value it is usual to assume a price for the metals.

CHAPTER XIII.

CALCULATION OF PROFITS.

Treatment Problems.—The profit that can be realized from any given ore with any given market price for metal produced depends on the metallurgical treatment and the working costs. Metallurgy has no place in this discussion, as it is an art of itself, a knowledge of which is necessary to the mine valuer. No matter how carefully a mine is sampled or how accurately the ore reserves are estimated, the whole work may be nullified by the application of the wrong method of treatment.

As a usual thing, the mine valuer is not an expert metallurgist, but it is essential for him to have a firm grasp of the issues involved and at least sufficient familiarity with the subject to indicate the correct method of treatment and the approximate cost thereof, even if he may not be able to carry out and take charge of the practical working of the method itself. No rules can be laid down for the guidance of the inexperienced—a knowledge of metallurgy is essential and each problem must have its own solution. The subjects on which the modern engineer is called to pass judgment are so varied that no man can be expert in all and it is a pity that more engineers do not recognize their limitations in regard to metallurgy, and before committing their clients to a heavy expenditure for plant, call in the metallurgist to direct them.

An analysis of the ore to be treated is not always a sufficient guide in deciding on the treatment and, wherever possible, large scale tests should be made to demonstrate the efficacy of the method to be used. Such tests will show the metallurgical losses and the percentage of the gross value that will be recovered in a marketable form. If a mixed lead-zinc ore is to be treated and each can be recovered as a separate product, the amount of such products must be determined and a knowledge of market conditions is necessary to enable the net value of each to be arrived at. Enough has been said to show that the gross metal contents of an ore is not necessarily an index of the net value and more especially is this true with ores consisting of a mixture of base metals, such as lead, zinc, and copper, with precious metals.

Working Costs.—The method of treatment having been decided on, the working costs can be determined. Through lack of skill and experience more errors are made in estimating costs than in any other branch of mine valuation, and more companies come to grief on account of under-estimated working costs than from over-estimated ore reserves. The short time an engineer spends on a mine examination in many instances enables him to obtain little accurate knowledge of the economic conditions of the district and an ample factor of safety must usually be allowed if serious errors are to be avoided. When data as to the costs at neighboring mines is available the problem is greatly simplified, but failing this, the engineer has only his experience to guide him, and in remote districts the problem may become one of great difficulty. It should be resolutely faced and, while care is advocated, it must not be carried to such extremes as to stifle a legitimate business.

The net value of the ore, or profit, is the difference between the cost of producing it and the actual net sum received from the sale of the metals or minerals recovered from it. What properly constitutes costs of operation is a question of accountancy, but from a valuation standpoint, it may be stated that whatever temporary disposition may be made of certain items of expenditure, when the mine is exhausted, the cost of production is equal to the difference between the amount realized from the sale of the products and the dividends distributed.

In the May, June and July, 1912, issues of the *Mining Magazine,* T. A. Rickard, in a series of articles, entitled 'Phantom Profits,' has drawn attention to the misconception that may arise from accepting ordinary statements denominated operating cost or working cost. Certain items of expenditure, such as extra remuneration to directors, cost of underwriting, head office expenses, and cost of extra equipment required by advances in metallurgy, etc., are not shown in the cost sheets but are allocated in the annual accounts. On the average these amount to 20% of the so-called operating cost. The instances quoted in the articles are all from operating companies, but they are equally important to the valuing engineer. Unfortunately, mine valuers make too little allowance for expense of that character.

The tendency during the past two or three decades has been for the price of all commodities, including labor, to go up. This is often overlooked and may lead to disastrous results. In new districts with a limited labor supply the starting of operations on a large scale may within a short time cause a serious rise in wages and thereby adversely affect working costs. These things must not be overlooked, nor must the likelihood of costs going up as greater depth is attained, on account of heavier pumping and winding charges. Another variable factor is that which may arise from variations in freight rates and the charges of customs smelters, where the output of a mine is shipped either as mined or in the form of concentrate. Freight rates may have a serious influence on costs where an ore is smelted locally and fuel or flux, or both, required to be brought from a distance.

Allowance should always be made for bad management, as the attainment of estimated working costs may be entirely dependent on highly efficient management. The cost of construction and development, as well as amortization of capital on the scale of operations proposed, can be fairly accurately established by the experienced engineer. Where he is likely to fail is in connection with those items which are purely of a head office character and which are entirely beyond his control.

Base Metal Prices.—In arriving at the value of any metal mine, except gold, the selling price of the metal must be taken into account. It is well known that the price of all the metals is subject to wide fluctuations, dependent on the law of supply and demand and other economic conditions, needless to discuss here. A study of the statistics is no guide to the future price and those most familiar with the business are just as likely to arrive at an incorrect figure as less experienced persons.

Mr. Hoover,* although recognizing the hazard in doing so, estimated in 1909 that from the available outlook the following would be the normal prices of various metals for "some time to come":

	Lead	*Spelter*	*Copper*	*Tin*	*Silver*
London £ per ton	13.5	21.	65	130	26d. per oz.
New York cents per lb.	4.3	5.0	14.	0.29	52c. " "

This estimate was made as the result of a careful study of the statistical position of the various metals combined with an analysis of the general trade conditions of the world, which naturally govern the demand for commodities of all kinds.

Despite Mr. Hoover's great ability and experience he failed to gauge actual metal prices in the four years which have elapsed for which the statistics are available. No man can successfully look any distance into the future. The demand for metals depends on trade activity and trade activity is governed by a number of factors, many of which are unforeseen, or even beyond control of man. Good harvests in various parts of the world will cause an expansion of all sorts of business, people have more money to spend, new construction is stimulated, new buildings, new railroads and other industrial enterprises are undertaken requiring larger amounts of the metals. On the other hand, drought or pestilence may have the opposite effect, unexpected political disturbances may restrict trade, which naturally affects metal prices in proportion. For this reason any forecast as to the future price of the metals, beyond perhaps a few months, is liable to be falsified by the attained figures.

The actual London prices per ton of the various metals for 1909, the year of Mr. Hoover's estimate, and the three succeeding years, is as follows:

	Lead *L. s. d.*	Spelter *L. s. d.*	Copper *L. s. d.*	Tin *L. s. d.*	Silver per oz. *d.*
Hoover.....	13/ 5/0	21/ 0/0	65/ 0/0	130/ 0/0	26.0000
1909.......	13/ 1/8	22/ 3/0	58/17/3	134/15/6	23.7217
1910.......	12/19/0	23/ 0/0	57/ 3/2	155/ 6/2	24.7085
1911.......	13/19/2	25/ 3/2	56/ 1/9	192/ 7/1	24.6531
1912.......	17/ 9/6	26/10/0	72/18/1	209/ 8/5	28.0349

The difference between Mr. Hoover's estimate and the actual average metal prices is interesting.

J. R. Finlay, employed by the state of Michigan, U. S. A., to value the copper mines of the state for taxation purposes a few years

*'Principles of Mining,' page 37.

ago, assumed as a basis for calculation, copper at 14c. per lb. In a recent utterance he admits a change of opinion and now believes 15c. will be nearer the average price in the future. These two instances are quoted with a view to showing how difficult it is for even those familiar with the subject to make any forecast as to future prices.

With the large percentage increase in the consumption of metals during the past few years it would look as if a great scarcity of copper must ensue. No new mines are being opened or equipped in the United States and every likely district has been thoroughly prospected. Elsewhere a similar condition exists, so that those most familiar with the copper industry predict higher prices for the red metal. The last eighteen months has brought to the front at least one great mine, namely, the Chuquicamata in Chile, but more important still is the successful application of the flotation methods for the treatment of copper ores. Tests so far made indicate an increased extraction of from 20 to 30% over that attained by water concentration. The introduction of this process in mines, like the Utah Copper, Nevada Consolidated, or other great producers, will immediately add considerable quantities of copper to the existing outputs.

The flotation process is applicable to a wide range of ores, and just as the cyanide process revolutionized the treatment of gold ores, so does flotation threaten to render available great quantities of material that heretofore would not lend itself to profitable handling. The history of the world shows that as the demand for a thing has arisen, the ingenuity of man has found a source of supply to meet it. Other factors of a similar nature affect the other metals and render a prognostication of their future prices difficult in the extreme.

An interesting table showing the 'Increase of World's Production of Metals,' prepared by Bedford McNeill,* is well worthy of study in this connection as emphasizing this difficulty:

INCREASE OF WORLD'S PRODUCTION OF METALS.

	1889.	1891.	1901.	1911.	Percentage Increase for Ten Years Ending 1911.
	Tons	Tons	Tons	Tons	
Pig Iron	....	....	41,000,000	65,000,000	58
Copper	261,205	279,391	526,000	884,000	68
Zinc	329,600	356,200	500,000	900,000	80
Lead	540,200	589,000	850,000	1,100,000	29
Tin	55,400	59,500	88,000	116,000	32
Nickel	1,800	4,700	9,000	24,000	144
Aluminum	70	328	7,500	46,000	513
Mercury	3,700	3,700	3,000	4,000	33
Silver	4,100	4,700	5,300	7,500	41
Gold	....	....	380	680	79
Antimony	....	....	10,000	23,000	130

The increased output of nickel, aluminum, and antimony is striking and is due to lower cost of manufacture and new uses for

*Presidential Address, Inst. Min. and Met., March, 1913.

those metals resulting therefrom. The increment with the more commonly used metals, while not so large in percentage, has been stupendous in point of actual quantity, namely, pig iron by 24,000,000 tons; copper, 358,000 tons; zinc, 400,000 tons, and lead, 250,000 tons. This represents in the aggregate a remarkable increase in the world's purchasing power, because taken over a period of years, prices have not fallen, but have, on the whole, risen. Within the ten-year periods themselves there have been fluctuations between wide extremes, and the average prices for individual years, when compared, bring home in a startling manner the necessity of making a forecast for the immediate future with some degree of accuracy. The average price over a term of years must be the guide in estimating profits, but the fact must not be overlooked that a few years of low metal prices may seriously cripple the finances of a company, especially in its earlier life, and may even be the cause of its bankruptcy.

It is not within the purpose of this book, nor is it essential, to fathom the economic causes underlying the increased cost of many commodities during the past few years; nevertheless, commodities do cost more, and safe to say no one could have forecast the percentage increase with any degree of accuracy. The engineer can only meet the problem that confronts him by using the evidence at hand and must not endeavor to peer too deeply into the dim haze of the future. A normal price of the metal must be assumed and, in view of the difficulty involved, it is better to be conservative rather than excessively optimistic.

During periods of high metal prices the average human being is more inclined to take an optimistic view of such matters than during a time of depression. One factor that is rarely taken into account is, that when metals are selling at high prices, giving greatly increased profits, development work is accelerated and other extraordinary expenditures undertaken, which, when metal prices are low, are entirely suspended.

The normal price or average price represents the mean between the high and low levels. Mr. Hoover sugests the idea that on the low swing metals reach a level below which the price will not fall and this he calls the basic price. It seems to me to be too difficult to determine such basic prices, and they must vary from time to time, depending on the law of supply and demand and other economic factors. One such factor on which the engineer cannot reckon with any degree of certainty is the result of those combinations of business interests known as trusts, pools, or conventions, which have for their object the regulation of prices. It is well known, for instance, that the price of spelter is artificially maintained, as is the price of antimony and some of the others to a lesser degree. With prices artificially regulated, wide fluctuations are less likely during the maintenance of the combination, but its dissolution, by permitting or even fostering competition, may result in prices below their legitimate level until accumulated stocks are exhausted.

CHAPTER XIV.

AMORTIZATION OF CAPITAL.

In the abstract, the total net profit contained in the ore reserves of a mine on a fixed basis for cost of production and prices realized by the metals produced, remains the same, regardless of the length of time it takes to realize that profit. On the other hand, the demands of finance require the consideration of interest rate on capital invested and, therefore, the time it takes to realize the profit in the ore reserves. This is governed by the number of tons per annum mined and treated.

Where interest rates are low, there is not the same necessity for intensity of production as where they are high. With a given amount of capital invested, the rate of production may be so low as to cause operations to be carried on at a loss, or at least not yield sufficient profit to pay interest on the investment, much less provide for a return of such capital. Increase the annual output sufficiently and the earnings increase to the extent required. This question of increase of the rate of production is closely interwoven with the size and capacity of the plant, and this brings in an additional financial factor, namely, the cost of such larger plant, which capital outlay in its turn must be amortized and return a suitable rate of interest. The plant may be increased beyond the capacity of the mine and, although the annual return may temporarily increase, the net result shows a smaller total than would have been achieved with a smaller plant.

The scale of operations necessary to afford an adequate return on the capital invested will vary with each different case, and the engineer must judge each one on its merits. A 2% copper ore under the conditions prevailing in most parts of the world can only be profitably handled on a scale large enough to warrant the expenditure of capital necessary to install labor-saving appliances of all kinds and the equipment required to secure low working costs. Unless the deposit to be exploited is capable of yielding the required tonnages during a period sufficiently long to return such capital invested, plus a satisfactory rate of interest, it cannot be considered commercially valuable.

The porphyry copper mines, such as the Utah Copper, Nevada Consolidated, Chino and others, where the copper occurs in a silicious gangue which requires to be separated from the valuable mineral by concentration, and the low-grade gold mines, such as the Alaska Treadwell and Homestake, are worked at a profit only because of the large tonnage handled. Were an attempt made to work these same mines with an output of, say 100 tons of ore per day, the inevitable result would be loss. The engineer must take this factor into his calculations as seriously influencing return of capital and interest on investment.

All capital is invested with the expectation that it will be returned together with a suitable rate of interest thereon. There is a wide gap between the methods employed, for instance, by the actuaries of a life insurance company in valuing its investments and those applied to mining. In the purchase of so-called gilt-edged securities, such as government bonds, a definite rate of interest, plus a return of the capital, can be easily calculated, and there is a practical assurance that such figure will be realized; the sole risk being the failure of the particular government to meet its obligations.

When we come to industrial enterprises a certain amount of risk is involved, due to the fact that the yield is more or less dependent on personal equation, varying economic conditions and other undetermined factors, which may influence the eventual profit. So far as the capital invested in such business is concerned, this is originally required for the purpose of equipping the business for operation, the purchase of plant and equipment, and so forth. The life of industrial enterprises is, in most cases, more or less unlimited, depending on the ability of each individual business to continue the struggle for existence by meeting competition, etc. Once the original capital outlay for plant and equipment is made, its cost can be amortized at a definite rate and, providing a suitable allowance is made for upkeep and renewals, the entire capital account can, if necessary, be liquidated in a predetermined number of years.

In making an investment in an industrial enterprise, the returns can be calculated with the same exactness as with gilt-edged securities, except that there is a greater risk involved, due to the inherent nature of the business. On the other hand, with a mine, every ton of ore that is extracted diminishes by a definite amount the assets of the business and brings the property so much nearer to exhaustion. An expansion of a mining enterprise is only a comparative statement and does not detract in any way from the fundamental fact, that with the exhaustion of the ore the business itself must cease. It is in this respect that mining differs from all other business and for that reason a mining investment must be viewed from an entirely different standpoint from the others. A great deal has been written about amortization of capital in mining enterprises, but its usefulness, except in a few special cases, is questionable.

It will appear from the foregoing, that in order to be able to reckon on amortization, it is necessary to have a more or less assured definite period during which the capital can be retired. In other words, the business must be assumed to continue for a sufficient number of years to permit of this. As has already been indicated, it is rarely possible to buy a mine where the net profit in the ore reserves is equal to the purchase price. The mine is bought on the basis of the proved, probable and prospective ore, and according to the amount of the purchase price represented by such prospective ore, to that extent does the element of speculation enter into the investment. Hence the difficulty of applying the rules of amortization to a mining enterprise. With such properties as the Rio Tinto, some of the porphyry coppers, the large Rand amalgamations and a few others, where the ore reserves are enormous, amortization tables may be applied, but in the vast majority of mines it is out of the question.

The theory of redemption or amortization of capital, according to Mr. Hoover, is that: "A portion of the annual earnings must be set aside in such a manner that when the mine is exhausted the original investment will have been restored."[1] This further involves that these annual instalments are considered as payments before the due date and, if placed at compound interest, will redeem the capital at the end of the period which in a mining enterprise would correspond to the time when the mine is exhausted. But Mr. Hoover observes that: "In the practical conduct of mines or mining companies, sinking funds for amortization of capital are never established."[2] The reason for this has already been given. I do not mean to convey the idea that amortization is not to be taken into consideration in valuing a mining property, but its real usefulness must be to serve as a guide to indicate the amount of risk involved. The rate of production naturally governs the annual income, and from this the number of years during which this must continue in order to amortize the capital and pay a suitable rate of interest thereon can easily be computed.

If the life of the mine, as determined by the proved and probable ore, is less than such term of years, then the difference is equal to the risk involved in making the purchase. On top of this the question naturally arises as to the amount of risk that it is advisable to allow in a mining investment. On this subject, too, a great deal has been written, but it cannot be indicated by any formula or any amount of printed matter. It must be decided by the engineer as the result of his study of the deposit and the known economic conditions. This involves broad questions of experience and judgment. A proper estimate of the geology of the deposit will permit of a forecast as to extension in depth, while the economic conditions involve such questions as varying metal prices, cost of labor, supplies and the like.

In the ordinary mining project that comes within the engineer's purview, there is rarely more than three or four years' ore in sight, and at the known rates of profit resulting from the operations at such mines, it becomes impossible to amortize the capital except in the manner already indicated, namely, by additions to the ore reserves or increased metal prices. If a mine can be purchased on a basis showing a full return of the capital, plus interest thereon, it would be an extremely desirable condition; unfortunately, however, this is rarely possible and we are, therefore, compelled to pay a portion of the purchase price for a possibility. In other words,

[1]'Principles of Mining,' Page 42.
[2]'Principles of Mining,' Page 42.

we cannot avoid the element of risk, which is almost inseparable from any mining transaction, and as Mr. Bedford McNeill states:* "It is no use ignoring or pretending to ignore the fundamental fact that mining is and must always continue to be essentially speculative." Therefore, it devolves upon the mining engineer to confine the risk within reasonable limits.

There are various ways of expressing the percentage of risk one is justified in taking. I believe J. H. Curle advocates that if the profit contained in the ore in sight is equal to 75% of the purchase price and the bottom of the mine is good, the purchase is justified. Others writers advocate a stated rate of interest per annum, but the application of any such rule is dependent on the particular conditions involved and attempting to make a law that can be generally applied is like trying to move a ship without raising the anchor—it is perfectly safe, but the ship will never get anywhere. Business acumen must remain the guide to what percentage of risk may be taken in any particular case.

*Presidential Address, Inst. Min. and Met., March, 1913.

CHAPTER XV.

WRITING REPORTS.

Every engineer has his own method of setting out the facts gathered in the course of his examination of a property, but nowhere is brevity more to be commended. In general, it may be stated that the report should give such data that another engineer, willing to assume the sampling as having been properly done, will be enabled to check the results and form his own opinion as to the value of the property in question.

It often happens, especially with younger engineers, ambitious to show the painstaking manner in which they have carried out their work, that the report is padded with verbose descriptions and discussions of useless topics. Instead, it should be confined strictly to such facts as influence the value of the mine, and all other topics, even those relating to mine management, had generally better be omitted, as it not only unnecessarily lengthens the report, but further tends to fog the vital issue. At times, it may be necessary to discuss managerial policy as affecting the profits to be derived from the business, but in nearly all cases this can better be discussed in an appendix.

There have been a number of report forms published giving the various headings and the order in which the topics entering into a mine report should be discussed. These are useful to some extent, especially to the younger engineer, but they must all be modified to meet the special conditions of each case.

The report should be as brief as consistent with an exposition of the facts and the topics should be taken up in logical sequence. One of the most common errors is, in the discussion of the geology, to anticipate the assay value of the ore or the tonnage available. No subject should be introduced until its proper place. Many reports contain pages of descriptive matter relating to the topography of the country, roads, water supply, etc., placed before the discussion of the mine workings proper. In most cases these should be added as an appendix to the main report, or such as influence the working costs should immediately precede the chapter dealing with that subject.

I recall a report written by a prominent engineer that consisted of more than 100 typewritten quarto pages of descriptive matter, that concluded by stating there were only 10,000 tons of 4 % copper ore available and the mine was not worthy of consideration. The property was situated in a remote district in the tropics, where skilled labor was not to be had and the other economic conditions adverse. The writer of the report described the country, its fauna and flora, the economic conditions, questions of climate, transport, labor and other things that might have an influence on a going concern, but were quite useless in view of the fact that the property was not recommended. All these things matter not a whit if the result of the examination shows the property to be of no present value or any likelihood of being so in the future.

Some engineers recommend that a report should open with a summary. This is illogical; the proper place for the summary and recommendations is at the end, where it is just as easily found. A report should be an impersonal document, and, as pointed out in Chapter I, should be an impartial exposition of the facts regardless of whether the engineer is acting for a buyer or seller. The engineer must not act as an advocate. Sometimes reports are used at a later date for a different purpose than originally intended. As a matter of personal protection and to prevent misconception on the part of the reader, and to prevent improper use being made of it the engineer should at the beginning of the report definitely state the object of the report and its scope.

The basis of all calculation of estimates should be definitely stated in unmistakable and concise language. The conclusions arrived at should likewise be definite. The engineer is called upon to give an opinion as to the value of the property and such opinion should be stated in unequivocal language. The custom is only too prevalent of qualifying recommendations in such a manner as to rob them almost entirely of their positiveness. The engineer's client has the right to expect a definite recommendation as much as a patient has a right to expect a remedy from his physician.

In foreign countries where a different system of weights, measures and currency is used, engineers are prone to employ both the expressions used in the foreign country and their own in the same report, even on the same page, so that considerable confusion may arise in the mind of the reader therefrom. For instance in a place like Mexico, where Mexican weights and measures are used, as also the metric system, together with Mexican dollars and gold dollars, a great deal of confusion may arise unless the same nomenclature is used throughout. Reports on Russian properties often lead to a great deal of confusion, because the engineer will use the Russian system in places and the English in others. A proper report should not only be built up logically, but the greatest effort should be made to convey a clear meaning. Engineers as a class lack literary ability, but this quality can be cultivated as well as any other.

The engineer is sometimes called upon to compile a report from second-hand data, that is, data which he has not himself gathered. This proceeding is very often fraught with a considerable amount of danger and may easily lead to serious mistakes being made. Naturally the basis for such a report must be the assumption that the information is correct. Very often this is not the case. No doubt the future operations of a mine are largely dependent on the quantity and grade of its ore reserves, and given a record of past operations one is generally justified in assuming a continuance of previously attained figures. In the estimation of ore reserves by the management of a mine the personal element enters more largely than it does in mine valuation work per se. In other words the mine valuer brings to bear a more judicial frame of mind than does the mine management, who may be influenced to a large extent by local conditions and prejudices, as well as having recorded certain opinions and figures in the past from which it may be difficult to withdraw. This in time may cause a certain amount of inaccuracy to creep into the reports of the mine management, rendering their use hazardous. A knowledge of the character and ability of the management will assist in a determination of the reliability of the data.

A mine may attain a given production and the estimates of the ore reserves may be correct; at the same time it may be difficult to continue outputting the figures attained in the past, because the mining work has not been properly done. It may be found that, in order to attain a high rate of production, the work of filling the stopes or otherwise supporting the working places has been neglected to such an extent that some of the largest and best producing stopes will have to be completely or partly shut down until the heavy ground is secured. Not only is the output affected by such a state of affairs, but naturally the working costs will have a tendency to be higher for at least a temporary period of time.

Another condition that is sometimes met in operating mines is that the management, in order to fulfill promises that have been made, will pick the eyes out of the mine, in other words, mine an excessive proportion of the best grade of ore with a view, of course, of maintaining a high rate of production. Many orebodies lend themselves to such a proceeding. Shoots may exist in the orebody which have a higher value than the general average tenor of the remainder, and if this better grade ore is drawn on, in too great a proportion, the time must eventually come when it will be exhausted and the remaining ore available for stoping will show a lower average tenor. In studying mine plans the limits of these ore-shoots are not always definitely located, so that unless great care is exercised in attempting to gauge the value of a property on second-hand data it may lead to an improper conclusion.

Recently I had occasion to check up a report by two well known German engineers, and the difference in the methods employed by them and that generally accepted as good practice by American and English engineers is quite worth discussing for the lessons taught thereby.

Here was a mine in which they estimated ore reserves amounting to more than 150,000 tons. This figure was determined by measuring with a planimeter the area stoped as shown on the mine plan, which was drawn on a scale of 1:1000, and correlating with it the recorded output of the mine. In this way a figure to represent the tonnage of ore per square metre of orebody stoped was obtained, and the unstoped ground was assumed to yield a similar tonnage, of course, of the same assay value. As a check on the smelter returns, about 20 samples were taken in various parts of the mine. The deposit was a fissure vein, averaging a little over 1 foot in width (35 cm.), and there were no vertical development openings on the orebody.

I do not think that good American or English practice would sanction the estimation of tonnage under such conditions. The history of the mine showed that about $450,000 had been spent on it, perhaps a great proportion of it uselessly, in excess of the actual return, and that during the latter portion of the period it was in active operation, before it fell into the hands of the mortgagee, the eyes were picked out of the mine.

In making this estimate of ore reserves, the engineers were accepting second-hand data. There was no real assurance that the surveys were properly made and that all the stoped areas were shown on the plans. It will readily be seen that on omission of any stoped ore would seriously affect the tonnage calculations and show a greater quantity of ore per square metre than was actually recovered.

The next great point of difference was in the manner of calculating the average assay value of the orebody. Our own practice demands, as has been explained in the preceding pages, that samples be taken at regular intervals, with an accurate determination of the width sampled. In the case under discussion, the engineers took the smelter returns in combination with the mine records of tonnage and calculated back that the ore averaged a stated amount, after allowing for losses in mining and concentration. The records of the mine showed that they had shipped arsenical concentrate and copper concentrate, which had been obtained from several different veins. A cursory inspection of the underground workings was sufficient to show that the principal orebody carried much more copper than other parts of the mine, yet the same average value for copper was given to this ore as to the remainder. The German engineers' figures showed an average value of 3% of copper. A number of check samples taken by me demonstrated an average value of between 8 and 10% of copper in the copper vein.

There is no doubt in my mind that this orebody could be mined and treated in such a manner as to yield a better result than the average given in the report. The mistake made and one in which we are concerned is that the existing methods of mining and concentration were accepted as efficient. Even assuming that the mining methods could not be very much improved upon, considering local conditions, the concentrator left a great deal to be desired. It was a most inefficient plant and the tailing probably assayed as much as the original ore. Further, this ore was a more or less massive sulphide and a very considerable percentage could have been hand-picked with better results than crushing the whole lot through a small mesh with heavy losses on inefficient tables. The acceptance of results of this kind is extremely risky and may lead to disaster.

The engineer examining a mine must be in a position to pass on the efficiency of the methods employed, and it is obvious in the case cited, that acceptance of second-hand information, such as that described, may lead into difficulties of all kinds. Here was a place where improved methods could easily be installed, to give a result entirely different from that shown.

In opposition of this, another case comes within my experience, which also refers to a mine on the continent of Europe, which was reported on by an engineer sent out from London by an English compay. It was a copper mine which had been in successful operation for nearly a century, with very much the same type of ore deposit as that described, but in this case, generally speaking, the proper methods of treatment had been adopted. Hand-picking was resorted to, to as great an extent as possible, in order to sort out the friable and valuable copper mineral in lumps, thus preventing the losses inevitable once it was crushed and submitted to water concentration. It is a well known fact that chalcopyrite, when finely crushed, has a great tendency to float and in water concentration serious losses result.

This property was bought by the English company. The engineer discarded the old method of treatment, designed a concentrating plant in which the whole of the ore was to be crushed to 10-mesh and decided to install two blast-furnace plants to treat the concentrate instead of using reverberatories as had properly been used all the many years the mine was in operation. Within 18 months the company had exhausted its capital and lost the property.

CHAPTER XVI.

SALTING.

The art of salting is probably as ancient as the art of mining itself. In the early days of mining in the western part of the United States, salting probably reached its highest stage of development. As the present time, however, it is not attempted so frequently, probably because we are better educated now and mine valuing is more scientific than it formerly was. There are probably 100 different ways of salting and 99 ways of finding it out, as a sage friend of mine very tritely puts it. Salting may be defined as the act of fraudulently increasing the value of a sample of ore for purposes of deception, although it may also mean any addition of valuable mineral to the sample beyond what legitimately belongs there, so that we may say a person is salting himself.

Of the two general classes of salting, one is to prepare the exposures beforehand and the other to tamper with the samples after they have been taken. The latter is the more customary method. At various places in these pages, attention has been called to the possibility of salting and the necessary precautions suggested. At no time during the entire course of the work should the engineer's vigilance be relaxed, not even after the sample is taken. It sometimes happens that great care is used during the whole of the underground sampling operations and caution thrown to the wind during the crushing operations or the assaying.

In the early days of mining, before mine valuation had attained anything like the scientific level it now occupies and when valuation was done by so-called practical men, who took a few samples underground in a perfunctory way, and when the more usual method of testing a property was by taking out sufficient ore to make a mill run, all sorts of roguery was indulged in and the literature of mining contains many illustrations of the clever manner in which the examining engineer was fooled, or attempts made to fool him. In those days breaking out samples by means of dynamite was not taboo as at the present time and a favorite pastime of the swindler was to load the dynamite cartridge with gold dust, which would, of course, be distributed with the ore on the explosion of the dynamite. Another favorite method was to salt the ground with gold dust inserted in the cracks of the rock or in vugs and the injection of gold solution in the samples after they had been taken. During mill runs it was an easy matter to add amalgam to the ore in the bins. In alluvial washing it is easy to spit tobacco juice containing gold dust into a sample, or for a man to conceal gold in his finger nails, which he liberated into the pan containing the sample for washing.

Mr. Simon* tells of an attempt to salt him in Siberia in the examination of an alluvial property. Repeated panning tests failed to reveal the existence of gold in profitable quantities, but at the earnest solicitation of the owners bulk tests were made. As the gravel had to be carted some distance to water, someone always accompanied the carts from the excavation to the washing place and great care taken that the sample was not interfered with during the washing operations, and notwithstanding this, each cart load revealed pay dirt. The fraud was not discovered until an empty cart was examined before filling, as no one up to that time had accompanied it on its return journey. This inspection revealed a harmless-looking cigarette paper containing the 'salt,' after which no more pay results were obtained.

An attempt was once made to salt me in the examination of an alluvial property. In various parts of the mine measured quantities of gravel were washed without revealing pay dirt. The owner was an engineer of good reputation; hence I was far from suspecting him. He insisted that there must be some accident, because his own work demonstrated that the ground developed contained sufficient gold to permit of it being worked at a profit, and he asked to be allowed to demonstrate this by putting several cubic yards through the washer. No objection was made and the gold was afterwards collected from the sluice boxes and yielded almost exactly the amount claimed by him. Unfortunately, however, among the gold collected from the sluice boxes was a nugget of peculiar shape that I identified as having seen among some specimen gold, which he had previously shown me, so I determined that I would stick to the results obtained by the panning. Not long after that the property was shut down.

The necessity for carefully guarding one's samples, after they have been sacked, can be illustrated by another story from actual practice. A firm in New York was asked by a client to interest themselves in a gold mine in the South. The engineer sent to make the inspection returned to New York and, on assay, his samples showed an average value of about $10 per ton. His principals were convinced from a long experience in Southern gold mining that, although nothing is impossible, at the same time it was highly improbable that any such orebody existed in the South. A re-examination by another engineer demonstrated the average value of the quartz to be less than $2 per ton. Upon investigation it turned out that engineer No. 1 had been presented by the seller with a sample of unusual occurrence of monazite sand containing gold. The sample was in a bottle about half empty, and the remainder had been used to salt his samples, as proved by panning. He afterwards remembered having left them lying at one part of the property while he went off somewhere else to take another

*Lecture, Diggers Club, London, 1913.

lot. *Don't let your samples out of your sight and even then be sure that they are securely tied and sealed with a distinctive seal and locked up in a mail sack or other safe receptacle.*

Perhaps the easiest thing to salt is gold ore, but even samples of copper ore have been salted by the addition of one or two small lumps of high-grade ore. In a sample running just under grade, the addition of a small lump of 60% copper ore may make the difference between commercial and non-commercial ore. To do this, the salter must get at the engineer's samples, but with the safeguards suggested there is little danger.

Only one other point need be emphasized, namely, that the examining engineer is responsible for his samples up to the very last and he must make sure that the preparation of the final sample for assay is done by responsible men. It would indeed be foolish to spend weeks in making a mine examination, if in the end the samples were turned over to an irresponsible person for reduction and assay. Local assayers interested in the camp are often dishonest and the 'back-door' assay, or the 'pencil' assay, is not unknown.

Sometimes an incompetent man will falsify his returns. As a case in point, some years ago in the examination of a tin mine in Tasmania where the tin occurred in a body of massive pyrite, I sent my samples to an assayer of good reputation in Melbourne, and the average of the results he returned showed the ore to contain about 2¾% tin. At the same time, one-fifth of the total number of samples were sent to another assayer to be checked, his results, arriving a few days later, averaged less than ½%. As the mine had produced considerable specimen tin and a great deal of alluvial tin had been washed from the creeks in the vicinity of the property, I was at first inclined to believe that the second assayer did not know his business, although he likewise was a man of good reputation and long experience. A third and a fourth assayer were called in, only to corroborate the low results obtained by No. 2. In the meantime No. 1 was interviewed and he insisted on the correctness of his results.

To settle the business it was deemed advisable to bring two half-ton lots from the property, which were crushed and quartered down by one of my assistants, and the resulting samples given a thorough test, with the result that it was demonstrated that the ore was not of commercial grade. It took two years to find out what had happened —assayer No. 1 in the meantime having gone out of business. Then I learned that, not being able to get the results he expected, and wishing to please me, he penciled in the results, his excuse being that he did not think that 2% or 3% of tin made any difference. This is one of the pitfalls that it is difficult to guard against, especially in cases where an engineer is at a distance from good assayers and must place himself in the hands of outsiders. However, if the precaution is taken to keep duplicate samples of the pulps and have them checked elsewhere, there is little likelihood of the engineer being fooled.

Sometimes as an additional safeguard, a few lumps of clean ore are taken and in the assay office these are carefully washed and assayed separately. Any serious difference between the results of these assays and the regular samples is sufficient cause for suspicion and investigation. Another subterfuge sometimes adopted is to have a few bags of waste rock mixed with the other samples and, if they are all salted alike, detection is inevitable.

CHAPTER XVII.

PROSPECTS.

The vast majority of engineers are called upon from time to time to examine mines that are only partly developed, so they may be classed as prospects. Prospects may be divided into two classes: (1) New discoveries; (2) re-opened old mines. I do not recollect ever having seen a definition of a prospect, but I presume the generally accepted meaning among the profession is, that a prospect is a mine that has no ore blocked out and whose value in consequence is entirely prospective. If we cannot calculate the existence of any tonnage of ore, we cannot calculate any net profit and the property can only have a prospective value. The question then arises, what price can the engineer's client afford to pay for the prospect, or is he justified in paying any? It is easy enough for the engineer to dodge the risk and make an adverse report, but in doing so he may deprive his client of a large possible profit.

New Discoveries.—No man can see into the ground, but some men can make a better guess than others, by which I mean to say, as I have already indicated elsewhere, that some men are closer observers than others and, therefore, can better read the indications that exist and in making a guess are more likely to arrive at the real position. Many a prospect is worth the expenditure of the money required for its development and, if the conditions warrant, the engineer is justified in recommending such expenditure. It must be understood by this, of course, that not only must the geological, mining, and economic conditions be taken into consideration, but also the financial requirements. In general, it may be stated that when the development openings on a prospect are in ore and bottom in ore, further expenditure may be recommended, if the other conditions are satisfactory.

Personally I do not believe that one is ever justified in recommending the expenditure of money where the development openings are in ore below commercial grade, except in the case of a copper property. In the latter class, as is well known, due to surface oxidation, the upper portions of the orebody may be leached of their copper contents and, until the zone of secondary enrichment is reached, commercial grade ore cannot be expected.

Re-opened Old Mines.—In the re-opening of old mines we are often confronted with false evidence. The old workings do not always tell the truth, nor are the deductions that one may reasonably make from the evidence they present to be depended upon except with great reservation. In examining the old workings of a mine we are inclined to take the width of the old stopes as prima facie evidence as to the width of the ore mined and, further, we are likely to start with the hypothesis that the attainments of modern times and the great improvements in mining and metallurgical methods will enable us to handle material that the old-timers could not possibly have treated at a profit. No greater fallacy ever existed.

Barring a few specific innovations, such as the cyanide process, the chief improvements that have been made within the last fifty years have been on the mechanical side, so that we are now able, nay compelled, to treat larger tonnages in order to obtain a profit. The old-timers were good miners and good metallurgists, but they only treated small quantities of material. Their concentration methods were good and their smelting was good. The great change that has been brought about is largely due to economic conditions. The purchasing power of gold has decreased so that comparatively the metals are not worth as much as they formerly were. As an instance we may cite the unsuccessful efforts of modern companies working the gold mines in Egypt. Labor is now much more costly than formerly and, in the days of low output and cheap labor, hand-sorting and other means of securing a high extraction was prevalent and cheap, so that the size of stopes is not necessarily an indication that values have been continuous for the entire width opened up.

Even at the present day in remote places we can find a survival of old-time conditions. A few years ago I visited a mine in Austria, which has been operating continuously for a century, and whose early history dates back to the Celts. A walk through the stopes would lead one to believe that the orebody had been 10 to 15 ft. wide, but, as a matter of fact, the stope was carried that width because of the existence of two streaks of high-grade ore, separated by material practically, if not quite, barren. The only possible index to this condition was the gob which had been thrown back into the mine as filling.

In examining an old mine, if all the old stopes are caved and inaccessible for inspection, it is quite easy to be deceived, and unless there is positive evidence at hand, the engineer is not justified in placing any great weight on the evidence afforded by the appearance of caved workings. I know of two copper mines that were re-opened and the old workings were extensive in each case. In one of them, assays of the old pillars showed ore running as high as 35% copper and yet development below the water level demonstrated the non-existence of ore in commercial quantities. The magnitude of the orebody, as indicated by the old stopes, was entirely falsified by the subsequent development.

Another point with old mines is the existence of dumps of waste rock. All through the states of North Carolina, South Carolina, and Georgia in the United States, are old mines on which in some cases are enormous dumps of white quartz—a very suspicious circumstance. Many of these old mines have been reopened, capital having been largely influenced by former State

Geologists' reports, and, speaking generally, it may be said that not in one instance has any profit resulted therefrom.

The existence of old dumps is unquestionably evidence that the ore occurred either as narrow streaks, as isolated bunches or as a stockwork, and that excessive amounts of waste rock had to be mined with it. As a matter of fact, the history of these Southern mines indicates that certainly in their later years they were not operated at a profit, but were run solely in the interest of a Stock Exchange clique, who manipulated the shares up and down on the discovery or exhaustion of any particular pocket.

It is much better to meet with the entire absence of waste dumps on an old mine, because this indicates that everything taken out was treated, and while this is not an absolute assurance that the property was at all times operated at a profit, yet it indicates an orebody of workable dimensions. A narrow orebody or one occurring in bunches, pockets, or similar form, will necessitate the extraction of some country rock with the ore.

CHAPTER XVIII.

A FEW SPECIAL CASES.

Miners as Samplers.—Sometimes an engineer reaches a mine with one or more assistants to find there is a great deal more work to be done than had been originally contemplated and, as the examination must necessarily be completed within a given period, he is compelled to seek outside assistance. It is usually practically impossible to get any technically trained assistants near the property, who can be relied upon and the only other alternative is for the engineer to make use of ordinary miners. This is, of course, open to the serious objection that such men are not trained samplers and may be of questionable loyalty; however, the average miner, through his familiarity with the tools, can in a short time be taught how to take a fairly accurate sample and, while the results obtained cannot be given the same dependence as the work of the regular staff, yet by close supervision sufficiently reliable results may be secured. The scratch lot should not be allowed to work alone but in company with one of the more trusted men, taking alternate samples, so that the work can be watched. After the completion of the sampling, the engineer or his assistants must take a number of check-samples as a check on the work done by the miners. Any factor of safety that is necessary can be applied and in this way the likelihood of serious error is avoided. Frankly, this procedure is open to many objections, but under the conditions described there seems to be no other alternative, and means the difference between getting a complete or even incomplete sampling of the mine. A part sampling means incomplete data and, therefore, may be the cause of an incorrect opinion, whereas, using the miners under proper supervision and check-sampling afterwards the satisfactory completion of the work in hand.

Checking Surveys.—In mine examinations, it is quite usual, on account of the difficulty of checking, to accept certain information as reliable. Particularly is this the case with reference to the depth of shafts or other data of a like character. Very often, especially with mines that have been in existence for a number of years, mistakes have been made in the surveys and the error has been carried through by successive surveyors. The distance between levels should always be checked, as the tonnage calculations are based on such measurements, and if second-hand data is accepted serious error may arise. I know of just such a case where a shaft was about 100 ft. shallower than it was generally believed to be. Many possible errors in surveying will naturally occur to experienced engineers and it is needless to multiply the illustrations here, sufficient to point out the necessity of checking all surveys on which estimates are based. In any case it must be apparent that many beautifully drawn plans have been prepared by incompetent surveyors and the engineer must take a sufficient number of important measurements to feel confident that the figures he has taken into his calculations are reliable.

Sometimes trouble arises from the fact that the examining engineer fails to make sure that the property he is examining is the one his clients have under option. Before much time is spent on the mine it should be the duty of the engineer to go round the boundaries of the mining claims and satisfy himself that the main openings are actually within the claims in which they are supposed to be. As a further guide, a combined surface and underground plan should be prepared for the purpose of actually showing the position of the underground workings with reference to the surface boundaries. As an illustration of the usefulness of this, I may quote an experience of my own that resulted beneficially to my clients, and that of another engineer whose neglect caused serious loss to his clients.

In the examination of the 'L' mine, I found that in portions of the property the vein would dip out of the side lines at about 300 ft. below the existing level, it was, therefore, necessary to peg out a number of other claims to protect the deep ground. Fortunately we were able to do this, as the ground had not been previously located. It can easily be seen that a case such as this might arise, where the ground desired was held under different ownership and unless satisfactory arrangements could be made the prospective value of the property would be definitely limited. I know of just such a case where the examining engineer did not take the precaution to secure the deep ground, or to warn his client regarding the necessity of it, with the result that after the purchase was completed the new owner discovered that the land covering the dip of the lode had been secured by a third person, and this eventually caused disaster.

Dressing Mines for Sale.—Vanity is just as apt to lead a mining engineer astray as any other man. The average expert is very prone to regard with contempt the ordinary rule-of-thumb operator. Oftentimes such a man is not given credit for being as astute as he really is, with the result that the engineer fails to judge the position correctly. An actual case of this sort is that of the 'B' mine which was floated a few years ago under particularly brilliant auspices, and had among its shareholders a number of well known mining engineers. The purchasers were deceived in the value of the property more by appearances than anything else. The mine unquestionably was dressed for sale, although I do not think that any misrepresentations were made by the owner, he giving the experts a free hand to carry out the inspection in any way they desired. Everywhere about the place, both surface and underground, things were in such a condition as to give an idea of bad management and slovenliness, but the owner in opening his mine sent to the mill only the easily-treated ore, leaving behind the refractory material ostensibly because his plant was not capable of treating low-grade material. It is true that the plant was not capable of treating the ore. The examining engineer believed that by introducing up-to-date methods of treatment, installing a first-class mill, putting the mine in proper order, supplying it with adequate hoisting machinery and other plant and in general introducing improved metallurgical and mining methods, he would be able to reduce the costs to such an extent as to be able to treat the ore at a profit.

The purchase of the mine was accordingly completed and after a great deal of money had been spent in equipment, the operations proved to be a failure. The engineer made a mistake. The mistake is one that many men are prone to make, namely, they backed their ability to make a profit by improving the methods of others. Sometimes all the factors are not taken into consideration, factors which are difficult to ascertain, largely because they are not so much scientific as technical. In the particular case under discussion, in checking working costs under the owner's management, the engineer overlooked the fact that the owner was able to operate more economically than a company, because he received no salary and on account of the small scale of his operations, he was able to fill several positions at once. Besides he was an extremely capable man, better than the average mine manager, with boundless energy, working for his own profit, and by practicing every economy, he was able to attain a working cost that could not be done under any other condition.

This is only one illustration of a number of a similar kind that are common enough in practice and are apt to fool any but the most experienced engineer, because it is difficult to bring ourselves to believe that actual figures, as shown by the books of a going mine, are not a true measure of the operations. T. A. Rickard[1] relates an experience somewhat of this character in connection with his examination of the Camp Bird mine, a number of years ago. His estimates of the tonnage and value of the ore reserves have been fully borne out by subsequent operations, but the working costs have been greatly exceeded, as he says because he neglected to take into account the inevitable higher expense connected with the operation of a mining property by a London company. It is only in rare cases that one is justified in assuming the ability to reduce working costs below those actually attained at the time of the examination.

Misrepresentation of Facts.—There are more fools than knaves in the mining business, but the engineer must not allow himself to be a victim of either class if he wishes to make a success of his profession. An engineer should gather his own data and not accept statements made by interested parties. He must guard not only against wilful misrepresentation, a practice not altogether unknown

[1]*Mining and Scientific Press,* May 24, 1913.

to sellers of mines, but also against misrepresentation due to ignorance. Even an honest man in a spirit of loyalty to his employer will put the best possible face on a business and may allow an examining engineer to be misled by the mere fact of his silence. There may be a question whether an engineer representing a seller, is committing a breach of professional ethics by withholding salient facts which the visiting engineer may have difficulty in ascertaining. An engineer must remember that a man selling a horse only talks of the horse's good points and leaves the buyer to find out the bad ones.

Some years ago I examined a prospect in the United States, and the manager in showing me over the ground pointed out what he considered the persistence of the ore through the three principal openings, which were not in a continuous line, but were at divergent angles to one another. There was very little work done, so I had considerable difficulty in sizing up the position. The manager was convinced in his own mind that these openings were all on the same deposit, but I could not make my observations fit in with any such theory. One of these three openings was an adit driven at the foot of the hill and had entered a few feet of ore. This gave the clue to the position, for at this cross-cut I was able to determine what I considered the strike of the lode and subsequent development work carried out on this hypothesis opened up a large tonnage of ore.

I know of a mine in Norway which was owned by two engineers, one of whom accompanied an examining engineer, representing a would-be purchaser. The length of the orebody was represented as its width. It is needless to say that a very brief examination satisfied the visitor as to this fact, and an unfavorable report was the result.

A case of wilful misrepresentation happened a few years ago in Australia, and the victim was a capable engineer. He was probably in a hurry and consequently did not take the trouble to crawl into and examine all the old workings, nor did he take the trouble to thoroughly inspect the ore exposures to which the most importance was attached. The old workings in the mine, where the principal work had been done, were opened by an inclined shaft from which three levels at short intervals had been driven a short distance along the strike of the lode. There were one or two other shafts from the surface which it was evident were in barren ground as they were stated to be off the lode.

A new vertical shaft had been sunk on the hanging side of the lode and at a depth of 200 ft. a short cross-cut struck the northern end of an orebody. The drift to the south was in ore, but the drift to the north was in a black, slick, apparently highly crushed material due to movement. The cross-cut itself was continued through a body of quartz, which was represented to be a cross-course which had caused the crushing of the schist in the north drift, but it was really the foot-wall of the lode, the hanging side being schist. A subsequent examination demonstrated these points, as well as the fact that from the old workings a connection had been made with one of the other old shafts, which the first engineer did not observe. Had he done so, he would have found that the connection was driven on the strike of the lode, but showed no ore, and that from the other shaft two cross-cuts were driven, one east and one west, without showing any ore. The second engineer arrived at the conclusion that the ore occurred as a cigar-shaped lens, the top of which was in the workings from the inclined shaft and the bottom of which was just touched by the cross-cut on the 200-ft. level from the main vertical shaft, therefore, there was no likelihood of opening up any large tonnage of ore. In other words, the mine was bottomed. Subsequent work by others demonstrated the correctness of these views.

The manager for the owners at the time of these examinations was interested in the sale of the property and, being unscrupulous, he did not hesitate to try and hoodwink the purchasers' engineer. The moral of this story is that, regardless of the inconveniences it may cause, every working in the proximity of an orebody should be inspected unless the examining engineer is willing to assume a negative value for such portions of the workings as are not visited. If an orebody has been opened up for a considerable length there may be some justification in assuming its continuance for some additional distance; where, however, only a small amount of development work has been done and there are other openings on the strike of the lode, these must be inspected for the information that may be obtained from them. If the first engineer had only taken the precaution to have some rubbish cleared away from in front of the connection above mentioned, he would have saved his clients considerable money and himself some loss of reputation.

CHAPTER XIX.

SPECIFIC GRAVITY.

Before any tonnage calculations can be undertaken, it is necessary to determine the specific gravity of the ore so as to arrive at the number of cubic feet per ton of ore in place. The accompanying table shows the specific gravity of various ores. The figures given are averages taken from Dana's Mineralogy and will be found close enough for ordinary practice. From the surveys and sampling measurements, the cubic contents of any block of ground may be ascertained and to get at the actual number of tons of ore in the block, the number of cubic feet per ton of ore in place must be determined. This is dependent on two things: (1) the specific gravity of the ore; (2) its compactness. There are several methods of determining specific gravity, which are set forth below.

The open spaces in the orebody, such as cracks and vugs, affect the actual weight of material that can be mined from a given volume of ground, so that after determining the specific gravity of the ore itself, an allowance must be made for these open spaces.

In addition to the ordinary methods of determining specific gravity by laboratory methods, occasionally it may be desirable for the sake of greater accuracy, to take out a given volume of ore and weigh it.

A suitable spot is selected in the orebody and a measured block of any given dimension of 1 to 3 or more cubic feet bulk is cut out and squared to dimensions by moil or chisel and the debris thus secured carefully collected and weighed. A simple calculation will then give the number of cubic feet of ore in place to the ton.

The following is taken from Furman's 'Manual of Assaying':

"The specific gravity of any body is the weight of that body as compared with the weight of an equal volume of another body which is assumed as a standard. The standard taken for solids and liquids is distilled water; for gases and vapors, dry air and occasionally hydrogen. All determinations of solids and liquids must be made at the same temperature. The temperature usually adopted is 60° Fahrenheit."

Of the various methods of ascertaining specific gravities of substances, the two here quoted are the only ones that need be taken into consideration in mine examination work:

1. Lumps.

Weigh first in the air, suspending the lump of ore from the beam of the balance by a piece of horse-hair, and then in distilled water whose temperature is 60° F. Let W = the weight in air, W′ = the weight in water, and Sp.gr. = the specific gravity; then

$$\text{Sp.gr.} = \frac{W}{W - W'}.$$

2. Fragments or powder.

Fill a specific-gravity bottle* with distilled water whose temperature is 60° F., and weigh it. This weight $=W'$. Weigh the substance in the air. This weight $= W$. Now introduce the weighed substance into the flask, fill it with distilled water, and weigh. This weight $= W''$:

$$\text{Sp.gr.} = \frac{W}{(W+W') - W''}$$

SPECIFIC GRAVITY OF MINERALS.[1]

Metal	Form	Mineral	Average Specific Gravity	Lb. Wt. per Cu. Ft.	No. of Cu. Ft. per ton 2000 lb.
Antimony.....	Native	....	6.7	417.8	4.8
	Sulphide	Stibnite	4.6	286.8	7.0
Arsenic.......	Native	Orpiment	5.8	361.6	5.5
	Sulphide	Realgar	3.5	218.2	9.2
Barium.......	Sulphate	Barite	4.5	280.5	7.1
	Carbonate	Witherite	4.3	268.1	7.4
Bitumen......	....	Carbon	1.5	93.5	21.4
Calcium......	Carbonate	Calcite	2.7	168.4	11.9
	"	Aragonite	3.0	187.1	10.7
	Sulphate	Gypsum	2.3	143.4	13.9
	Fluorite	Fluorspar	3.2	199.4	10.0
	Phosphate	Apatite	3.2	199.4	10.0
Coal.........	Anthracite	....	1.5	93.5	21.4
	Bituminous	....	1.3	81.0	24.6
	Lignite	....	...	.....	....
Cobalt.......	Sulphide	Linnaeite	4.9	305.5	6.5
	Co-Ni-Arsenide	Smaltite	6.8	424.0	4.7
	Co.As.S.	Cobaltite	6.2	386.6	5.2
	Arsenate	Erythrite	3.0	187.1	10.7
Copper.......	Native	....	8.9	554.9	3.6
	Sulphide	Chalcocite	5.7	355.4	5.6
	Cu–Fe–S	Chalcopyrite	4.2	262.0	7.6
	Cu–Fe–S	Bornite	5.0	311.8	6.4
	Cu–S–As	Enargite	4.4	274.4	7.3
	Cu–S–Sb	Tetrahedrite	4.9	305.5	6.5
	Oxichloride	Atacamite	3.8	236.9	8.4
	Oxide	Cuprite	6.0	374.1	5.3
	"	Melaconite	...	.....	...
	Sulphate	Chalcanthite	2.2	137.2	14.6
	Carbonate	Malachite	3.9	243.1	8.2
	"	Azurite	3.7	230.7	8.7
	Silicate	Chrysocolla	2.2	137.2	14.6
	"	Dioptase	3.3	205.7	9.1
Gold.........	Native	....	19.0	1184.7	1.7
Iron..........	Sulphide	Pyrite	5.1	318.0	6.3
	"	Marcasite	4.8	299.3	6.7
	"	Pyrrhotite	4.6	286.8	7.0
	Arseno-sulphide	Arsenopytite	6.0	374.1	5.3
	Oxide	Hematite	5.0	311.8	6.4
	Titanic Oxide	Menaccanite	4.8	299.3	6.7
	Oxide	Magnetite	5.0	311.8	6.4
	"	Limonite	3.8	236.9	8.4
	Carbonate	Siderite	3.8	236.9	8.4

*If a specific-gravity bottle is not at hand. take a thin glass flask with a narrow neck and scratch a mark on the neck. The flask is to be filled to this mark in the determinations.

[1]Average figures are used, based on those in Dana's Mineralogy.

SPECIFIC GRAVITY OF MINERALS—Cont.

Metal	Form	Mineral	Average Specific Gravity	Lb. Wt. per Cu. Ft.	No. of Cu. Ft. per ton 2000 lb.
Lead.........	1. Sulphide	Galena	7.3	455.1	4.4
	2. Carbonate	Cerussite	6.5	405.3	4.9
	3. Sulphate	Anglesite	6.4	399.0	5.0
	4. Chromate	Crocoite	6.0	374.1	5.3
	5. Phosphate	Pyromorphite	7.0	436.4	4.6
Manganese....	Dioxide	Pyrolusite	4.8	299.3	6.7
	"	Psilomelane	4.2	262.0	7.6
	"	Wad	3.5	218.2	9.1
	Carbonate	Rhodochrosite	3.6	224.4	8.9
	Silicate	Rhodonite	3.6	224.4	8.9
Mercury......	Native	....	14.4	897.9	2.2
	Sulphide	Cinnabar	9.0	561.1	3.5
Molybdenum..	"	Molybdenite	4.6	286.7	7.0
Nickel........	"	Millerite	5.6	349.2	5.7
	Arsenical	Niccolite	7.5	467.6	4.3
Platinum.....	Native	....	17.5	1091.2	1.8
Silver........	Native	....	2.8	174.6	11.5
	Sulphide	Argentite	7.3	455.1	4.4
	Telluride	Hessite	8.5	530.0	3.8
	Ag–Au–Te	Petzite	9.0	561.1	3.6
	Ag–Au–Te	Sylvanite	8.0	498.8	4.0
	Ag–Sb–S.	Pyrargyrite	5.8	361.6	5.5
	Ag–As–S.	Stephanite	6.3	392.8	5.1
	Ag–As–S.	Proustite	5.5	342.9	5.8
	Chloride	Cerargyrite	5.4	336.7	6.0
Sulphur......	Native	....	2.1	130.9	15.3
Tin..........	Oxide	Cassiterite	6.8	424.0	4.7
	Sulphide	Stannite	4.5	280.5	7.1
Tungsten.....	Fe–Mn–W	Wolframite	7.3	455.1	4.4
	Ca–W.	Scheelite	6.0	374.2	5.3
Zinc..........	Sulphide	Blende	4.0	249.4	8.0
	Oxide	Zincite	5.7	355.4	5.6
	Sulphate	Goslarite	2.0	124.7	16.0
	Carbonate	Smithsonite	4.4	214.4	7.3
	Silicate	Calamine	3.7	230.7	8.7
Rocks		*Composition*			
Quartz.......	....	Silica	2.6	162.1	12.3
Andesite...... Basalt........	....	....	2.9	181.0	11.1
Diabase...... Diorite.......	....	....	3.0	187.0	10.6
Granite.......	....	....	2.7	168.0	11.9
Limestone....	....	....	2.7	168.0	11.9
Porphyry.....	....	....	2.73	170.0	11.8
Rhyolite...... Sandstone....	....	....	2.4	149.6	13.4
Schist........	....	....	2.7	168.0	11.9
Shale.........	....	....	2.6	162.1	12.3

CHAPTER XX.

THE SAMPLING OF PLACER DEPOSITS

By Chester Wells Purington.

Preliminary.—In the following chapter, which relates to the sampling of alluvial deposits only, I am obliged to make the barest reference to the great subject of the cost of placer mining. The costs of dredging, of hydraulicking, of hydraulic elevating, and of drift or channel mining, are of an intricate nature, and their analysis necessitates a treatment of many component elements. No placer engineer is qualified to make an examination of a gravel deposit who has not a thorough knowledge of the cost of gravel excavation and alluvial metal saving, as exemplified in the principal placer districts of the world. Volumes have been written on the cost of gold dredging in various countries, and yet in every new case, the situation and environment of the particular property which the engineer has to examine will control the cost. To know the gross recoverable value per cubic yard of a given mass of gravel, be it ever so rich, is of no use unless the engineer has in his possession that knowledge of previous operations under similar conditions which will allow him to form a conservative estimate of the cost of extracting and marketing the contained metal and getting the money returns for the same.

Placer operations, which deal as a rule with much larger quantities of material than do lode operations, frequently necessitate a proportionately heavier expenditure for preparation and equipment. This expenditure as well as purchase price must be entirely redeemed in the ordinarily short life of the placer mine.

The engineer should not fail to allow for every possible item of working and redemption cost and should not, for the sake of false economy, base his estimate for equipment on the use of cheap and futile machinery. Many a spectacular hydraulic mine running today is a costly luxury to the owners and, notwithstanding the numerous successes attained, many a gold dredge lies stranded in the grassy valleys of California or Colorado, and on the wind-swept tundras of Alaska.

For the sake of uniformity all references to gold values are made in terms of U. S. currency, which is standard in placer sampling, And it should be further said that all weighings of placer gold and platinum samples are done in milligrams. Placer gold ranges from 650 to 950 fine and varies with each locality. One gram of pure gold is worth $0.6646. A safe value for all ordinary cases is to count 20 milligrams to one cent, or 1 mg. as worth 0.05 of 1 cent.

The only instance in which the chemical assay is allowable in connection with placer sampling is in the case of alluvial tin deposits where a heavy concentrate is obtained in the pan, consisting of fine cassiterite, magnetite, wolframite, garnet, rutile, etc., impossible to separate mechanically in the field. As this concentrate represents the commercial product obtainable in the sluice-box or on the dredge table, the sample must be submitted to chemical assay before the value of the concentrated product, which it is mechanically possible to save, can be arrived at.

It is, however, an extraordinary fact that reports on alluvial gold properties are occasionally submitted with lists of samples which have been submitted to fire assay. Any engineer who would fire-assay the samples from a gold gravel mine might be retained to make one report. He would probably never have the opportunity to make another one.

General.—The procedure in sampling gravel deposits in the majority of cases with which the engineer has to deal, approaches much more nearly to mathematical exactitude than does the sampling of deposits *in situ.* However extensively an original mass of ore in place, be it a vein, ore bed, or impregnation, may be developed, and however thoroughly the examining engineer applies those laborious methods of determining the value which have been so ably described in the foregoing pages, there is always in such cases an unknown factor which concerns the extension of the ore-mass in length or depth beyond the exposed faces. This unknown factor, in the case of gravel deposits either is non-existent or it exists in a very subordinate degree. Broadly speaking, the value of a placer deposit containing a workable quantity of an economic mineral depends on its area or horizontal extension, while the value of an ore deposit depends on its extension in depth (vertical or inclined).

Placer deposits which are too deep to be economically drilled or tested by prospecting shafts, are seldom of importance, except in the case of deep-lying channels of gravel covered by ancient lavas. Just so far as gravel or placer sampling calls for the exact methods of the mathematician, or civil engineer, in the same measure it cannot be said to allow room for the exercise of that fine intellectual and nearly indescribable quality called judgment, to which attention has been so emphatically called in the preceding pages, as an essential qualification of what may be described as the hard-rock mining engineer. While it is gratifying to the placer engineer to know in advance that he will in every case deal with predetermined knowable quantities, so that there is no room for a guess, he knows, too, that his work is far less interesting; that it is on a lower intellectual plane and approaches more nearly to the rule-of-thumb methods of the craftsman, than does the intricate duty of the engineer engaged in sampling a large deposit *in situ,* which last entails a fundamental knowledge of the great subject of ore deposits, keen judgment, wide metallurgical training, and in addition an ability to apply the precise methods essential in all engineering practice.

Because the bulk of valuable metal won from alluvial deposits is gold, and because an increasing quantity of gold is won every year by the process of dredging, the present-day placer engineer, in nine cases out of ten, is called on to sample gold gravel deposits suitable, or assumed to be suitable, for gold dredging.

The reduction of the methods of sampling gold-dredging areas to a scientific basis is a development of the last fifteen years, and is thus a much more recent acquisition to engineering practice than the sampling of underground ore workings. There have come to be recognized certain well defined methods, one of which the engineer must choose, if possible, in accordance with the character and situation of the gravel deposit in question. He must also recognize that certain attendant conditions, such as the nature of the bedrock, the presence of boulders, and excessive amount of clay, too great or too shallow depth of ground, insufficient water supply, or other disadvantages entirely apart from the gold contents of the gravel, may justify an unfavorable opinion of the property, irrespective of results obtained from the pit-sinking or drilling returns. These special accessory conditions are as important as the gold tenor to the success of a dredging enterprise.

Classes of Deposits.—Before describing the sampling methods, a brief description and classification of gravel deposits will be given, as, notwithstanding the present and increasing interest in the gold-dredging industry, important and highly productive deposits of gravel exist which must be sampled when opportunity arises, with a view to exploitation by entirely different methods.

The term 'gravel deposit,' as used in this chapter, includes only those detrital deposits which may be exploited for their valuable metallic contents and are referred to as placers or alluvial deposits. This naturally excludes all deposits of gravel which are worked merely for use as road metal, railway ballast, in concrete work and the like. Placers include, according to a decision of the United States Supreme Court, all deposits in which a valuable mineral occurs in the softer material which covers the earth's surface, as distinguished from occurrences in the rocks beneath.

As the mineral sought sometimes lies in a matrix of fine sand, or even entirely in the crevices of the bedrock on which the alluvial deposit rests, the term, 'gravel deposit' is obviously not always applicable. Likewise, in the case of alluvial deposits, or those in which the metalliferous layer is a result of concentration from the rotting of rock and veins in place to a greater or less depth, the material cannot be called gravel in the strict sense of the word; therefore, it is preferable to use the simple word 'placer' to describe all valuable alluvial deposits, which may be exploited by the various forms of surface mining. The only kind of alluvial deposit which cannot be properly called a placer is that which lies too deep to be profitably exploited by surface cuts, as, for example, the Tertiary basalt-covered gravel channels of California and Australia or the deep channels of Fairbanks, Alaska, or Bodaibo, Siberia, covered with frozen silt over-burden and, for the sake of simplicity, these will be described by the term 'buried placers.'

The valuable metals sought in placer deposits are in the order of their importance—gold, tin, platinum, monazite, and the metals of the osmiridium group. Placers containing diamonds, rubies, and other precious stones, are of such rare occurrence that they need not be considered here.

Gravel deposits may be classified as follows:

1. Buried placers—Deposits buried under from 75 to 1,000 ft. of overlying alluvial or lava.
2. Creek placers—Placers in, adjacent to, and at the level of small streams.
3. Hillside placers—Placers on slopes, intermediate between creek and bench claims.
4. Bench placers—Placers in ancient stream deposits from 50 to 300 ft. above present streams.
5. River bar placers—Placers on gravel flats in or adjacent to the beds of large streams.
6. Gravel plain (tundra) placers—Placers in the coastal plain of Seward peninsula.
7. Seabeach placers—Placers adjacent to seashore to which the waves have access.
8. Lake bed placers—Placers accumulated in the beds of present or ancient lakes; generally formed by landslides or glacial damming.

These eight classes of deposits will be considered in sequence, from the standpoint of their respective adaptability to different methods of taking the samples.

The method of washing the material obtained in the sample will be considered later.

Class 1. Buried Placers.—These must be worked in almost every case by underground or drift mining, and, as the engineer is so rarely called on to sample such a deposit, the subject may be dismissed with a few words. Should the deposit be covered by 100 feet or more of the over-burden, whether it be basalt or silt and overlying barren gravel, six-inch drills, operated by steam or gasoline power, must be employed in the vast majority of cases. Rarely a so-called channel or drift mine is for sale in California, where working faces on the valuable bed of gravel are available for sampling. Such, for example, was the examination of the Red Point mine in Placer county, California, several years ago.

Certain drift mines in Australia and in the Lena and Nerchinsk districts of Siberia, where a heavy barren alluvial covers the pay gravel from 100 to 250 ft. thick, have been worked for long periods and may, when offered for sale, present certain blocks of gravel in the form of thin horizontal sheets, opened by tunnels on three or four sides. A thorough sampling of the Industrial Company's property in the Lena district of Siberia was made in 1897-98 by two American engineers, samples being broken from the faces exposed underground, in much the same manner as would be done in the case of a horizontal bed of lead or zinc ore in limestone.

Class 2. Creek Placers.—It is seldom that an engineer is called on to sample a small creek deposit. In doing so he will bear in mind that the gold will probably be coarse and irregularly distributed and that much water will be present, in all probability preventing sampling by pits. General conditions will favor close sampling by four-inch hand drills, as topography is likely to be steep, and as workable creek deposits occur only in new and uncivilized countries. Roads will be few and transport of heavy power drills difficult.

Under the class of creek deposits, comes the subordinate class of deposits formed by decomposition of rock in place, only slightly concentrated by stream action. It is known in advance by the examining engineer that values in such a deposit are extremely irregular and minute care is necessary in sampling. Examples of this type of alluvial deposit are found in Nigeria, where the metal sought is tin, and in Eastern Russia, where such deposits have been worked for gold for nearly a century. The well known 'saprolite' deposits which were at one time highly productive in Georgia and North Carolina, U. S., were largely of this type.

Class 3. Hillside Placers.—These occasionally form the lateral extensions of creek or river bar placers, the whole together forming a sufficiently extensive mass of gravel to make exploitation by gold dredging or by underground drift mining, a possibility.

The characteristic of such deposits is an approximately level cross-section of the bed rock extending from wall to wall of the valley. The over-burden gets progressively deeper approaching the sides. Deep drilling with power drills is the only satisfactory way to sample such a deposit.

Class 4. Bench Placers—Include those rare but eagerly sought gold alluvials which are suitable, providing water supply conditions are right, for hydraulic mining. The distinguishing characteristic is the level of the sill of rock on which the gravel rests relatively to the level of the water of the existing stream, contiguous to which the deposit lies. The difference between these levels constitutes the 'dump room,' in other words, the vertical height available for the tailing from the hydraulic sluices. The classic occurrence of bench placers is the Tertiary river system of California, which has been elevated and dissected by present streams so that at times the cañons of the existing streams lie hundreds of feet below the rock sills or shelves on which the ancient gravel beds were deposited.

The thickness of ancient benches from top to rock floor varies between wide limits. Perhaps the commonest thickness which the engineer is called on to sample is from 30 to 100 ft. Sampling is difficult and expensive, and the choice of the method of sampling depends on many conditions.

Pit-sinking is generally impossible on account of depth. If the bench is intersected longitudinally by a present stream, so as to expose it in section, side creeks may be taken advantage of to ground-sluice a certain amount of the gravel, thus giving at intervals a representative cut from top to bottom, from which a sample of 50 to 100 cu. yd. may be washed. I have in mind a gravel bench 300 ft. thick, and over 2 miles long, in Alaska, which was sampled in that manner. Drilling with hand drills may be practicable. If the gravel bank is less than 50 ft. in depth, and contains no large boulders, this method is preferable. The use of power drills may be imperative, and in this case, preparation may be made and expense allowed for drilling a series of holes, from 100 to as much as 400 ft. in depth, the cost of which may be from $500 to $1,000 each.

It should be said that the examination of large hydraulic properties by drilling is rarely called for, as such mines have generally been started by individual miners, and a working option is given to intending purchasers, who are content to operate the property on a gradually increasing scale until it is either self-supporting or abandoned as too poor to pay. A set of conditions fully as important to the success of hydraulicking bench deposits as the gold content of the gravel, relate to water supply, grade of streams, and perhaps most important of all, dump room.

Classes 5 and 6. River Bar and Gravel Plain Deposits.—These are not divided by any sharp line. Generally they lie adjacent to creeks and small rivers and in them the bulk of the world's alluvial gold is distributed, although rarely of sufficient richness to pay to mine. Deposits of this character present the best physical conditions for dredging, but it is rare indeed that the gravel is valuable enough to pay. They are generally of great lateral extent, sometimes several miles wide, and vary in depth from 15 to 75 ft. and even more. They may consist of gravel from surface to bedrock or may be covered with an over-burden of soil or peat.

Dredging is the only method by which they are exploitable, and sampling in most cases must be done by drilling, although pit-sinking is preferable wherever possible. Conditions in these deposits are generally eminently suitable for drilling with 6-in. power traction drills of the Keystone type. The ground is approximately flat, with a hard surface on which the drill can move itself easily from place to place.

Class 7. Seabeach Placers.—This class is so limited that it scarcely merits consideration. In only a few places in the world have such deposits been known to pay the expense of mining, the most notable being the present beach at Nome, Alaska. They are the resort of individual miners, who operate generally with simple rocker or sluice box.

Class 8. Lake Bed and Harbor Deposits.—This very limited class of alluvial deposits is rarely payable. Gold in lake beds is known and recently tin-bearing alluvial has been profitably exploited in harbors in Siam and similar deposits are known in Cornwall. To sample such a deposit, either a hand or power drill, mounted on a scow or barge, should be used. The fine character of the alluvial detritus generally will render the drilling comparatively expeditious and inexpensive.

Characteristics of the Pay Layer.—The gold or other valuable constituent is generally associated with valueless magnetic sand in the bottom layer next to bedrock, and sometimes entirely in the bedrock crevices. In typical so-called gravel wash, the pay gravel is ordinarily recognized by certain local characteristics, or indications, such as a blue or yellow color, the presence of binding clay, relative coarseness of material, or the like. While black or magnetic sand is sometimes entirely absent, as a rule the pay streak contains a far greater amount of oxide of iron than any overlying layer. This is frequently accompanied or even entirely replaced by garnet sand, the so-called 'ruby sand' of the miner. When there are igneous rocks in the vicinity other heavy materials, such as spinel, zircon, or rutile, are also found with the gold.

Origin of Auriferous Plains.—A short description of the origin of those gravel deposits which present physical conditions favorable to dredging follows. In the formation of alluvials the wearing down of mountains, consisting of rocks penetrated by auriferous veins, causes the formation of valleys of varying width, depth and length.

When these valleys are of small length but great width, they indicate that stream action has been rapid, and the decomposition and distribution of the detritus have gone on with proportionate rapidity. These are favorable valleys for the concentration of alluvial gold. On the other hand, valleys extremely long, narrow and of comparatively shallow width, are an indication that erosion has been slow and streams sluggish. The gold will travel a comparatively short distance from its source and will be distributed in narrow and short pay channels. Consequently such regions do not give promise of economic alluvial conditions.

The controlling factor is the amount of elevation to which the land was originally subjected. Comparatively intense and protracted elevation has occurred in California, and far north to the Yukon, resulting in such benches as occur at Georgia Hill, in the Forest Hill divide, and in the White Channel of the Klondike. On the other hand, in regions presenting the topographical development of the northern Altai in Siberia, the comparatively narrow valleys retain the same width for great lengths, and are evidently the result of processes of extremely gentle uplift followed by necessarily slow degradation.

The fundamental geographic dissimilarity between these two conditions and the processes resulting therefrom have a considerable influence in determining whether a given mass of auriferous detritus will prove payable or unpayable. In other words, granting that the amount of gold originally contained in equal masses of eroded rock was equal, the manner of its subsequent distribution will, in the case of the short, wide and rapidly eroded valley, be a factor in favor of a profitable deposit, while in the case of the long, comparatively narrow and slowly carved valley the physiographic causes militate against the presence of an extensive sheet of profitable gravel.

Distribution of Pay Channels in the Plains.—Whether or not the topographic conditions have been favorable to the occurrence of an extensive deposit, it is almost an invariable rule that the most abundant and coarsest gold will occur in defined channels as a result of stream deposition. Considering a gravel plain deposit, as the type most frequently to be examined, the course of the present stream which meanders through such a plain sometimes affords little index of the actual course, width, or situation of the valuable gold-bearing channel which lies beneath.

Fine colors or particles of gold may or may not occur in the surface alluvial, but this gold is merely the result of annual distribution and re-distribution by floods and currents which disturb the upper layers of fine gravel, but do not touch the bedrock pay. A preliminary line of pits or drill holes directly across the valley will quickly discover if any pay-channel exists below the flood plain and also determine its situation and width. The definition of what constitutes pay gravel naturally varies in each separate case, according as its recoverable tenor in gold, or other metal does or does not leave the required margin of profit above the estimated cost of production. The entire gravel plain of a valley, for say a mile in width and five miles in length, and a thickness of thirty feet, may contain gold, but the pay-channel width and length in this plain may be of such insignificant proportions that the property has no commercial value.

Preliminary Remarks to Sampling.—The placer engineer generally has an indication in advance as to what character of deposit he is expected to sample. If it is to be a dredging property, it is unlikely to have been previously worked. He should be on his guard, however, both in the case of dredging ground, and of hydraulic benches, to note how much of the bottom pay gravel has been drifted. The term 'drifting' in alluvial mining is applied to a system of underground gravel mining, similar to the long wall system used in a coal mine.

The placer deposit which is for sale is rarely one where actual work is in progress, or has been done for the purpose of development. Sampling of *in situ* deposits can be done only after a considerable amount of development work has been accomplished previous to the visit of the engineer. The gravel sampler, on the other hand, has to block out the reserves by the very process of sampling which he undertakes, so that it is nothing more or less than prospecting, or dealing with quantities which are unknown.

As a corollary to this, and this point is extremely important, it should be remembered that when a gravel property has been completely sampled, it has also been completely developed. In other words, as one is dealing with a horizontal instead of a vertical or nearly vertical sheet, the gravel examination places the whole life of the mine in sight, because its limits can be actually determined, while the examination of *in situ* deposits can only refer to that portion of the life of the mine, represented by the ore opened up by the existing workings. In other words, with the placer deposit there is no prospective ore. These two operations interlock or overlap in the case of disseminated copper deposits and horizontally bedded deposits where the number of beds is definitely known.

Conditions Affecting Value of Property.—As in the case of ore-bodies *in situ,* the engineer must immediately investigate the local conditions controlling the cost of working. Defective titles, difficulties of transport, inferior quality or insufficient supply of labor, insufficient water supply, shortness of season due to climate, or any one of a number of adverse conditions may warrant condemning the property without any drilling whatsoever. In such a case, the engineer saves money for his client by immediately leaving the property, notwithstanding the fact that a heavy expense may have been incurred in preparations.

Instruments and Equipment.—Besides the necessary special equipment of tools and implements, the following essential implements should be provided in any placer examination:

Aneroid barometer.
Brunton compass.
Level and rods.
Drawing implements, paper, and note books.
Carpenter's pencils.
1 steel tape.
1 linen tape.
Blocks of drill and time logs, good for both shafts and drill holes.
1 two-foot rule.
100 small, strong, 1 by 3/8 in. bottles with corks.
500 sheets light, strong bond paper.
1 powerful pocket lens.
1 balance sensitive to 1/2 mg. and weights with no glass part.
6 gold pans of steel, not (thin) iron.
3 lb. quicksilver.
Annealing cups.
Nitric acid.
Alcohol lamp and alcohol.
Magnet.
Concentrate dishes.
Gummed labels.
2 dozen 6x8-in. canvas sacks.
1 sheet canvas.
1 prospector's pick.

1 short-handled shovel.
3 galvanized iron panning tubs 3 ft. in diameter by 18 in. deep.
1 retort stand, alcohol, and stove for same.
Panning tubs of wood or galvanized iron.

If several panners are to work under one chief engineer, many of the above will be duplicated.

Rocker and Clean-up Boxes.—A most essential implement is the California rocker, which is shown in Fig. 22. It may be made smaller than this in proportion, but I have found this size to answer for all general purposes. Working dimensions for constructing this rocker are given by W. H. Radford in the *Mining & Scientific Press* of June 8, 1912. This may be made on the ground if carpenters are available; otherwise it must be made beforehand and transported to the ground in knocked-down condition. A dipper (which may be made out of a 2-quart tomato can) with a handle is also necessary.

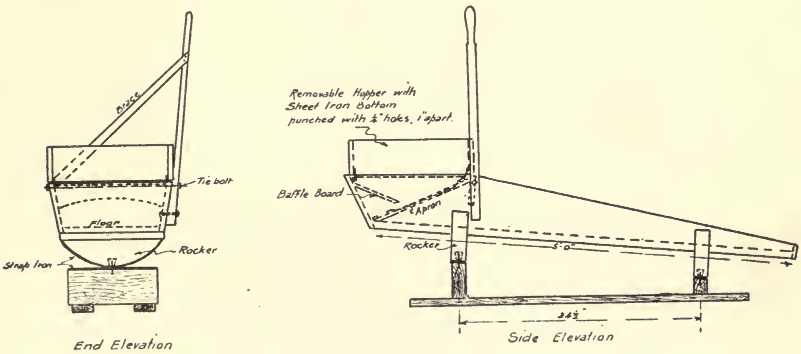

Fig 22. CALIFORNIA ROCKER.

If the hand drill is to be used, a special clean-up box must be made about 12x18x12 in. deep, standing on legs about 1 foot above the ground, open at one end, and fitted with a gate, into which the contents of the sand-pump is dumped and under which the pan is placed. If the power drill is used, large and strong clean-up boxes must be made for catching the pulp drawn up with each pumping of the sand-pump.

Hand Pumps.—If shaft-sinking is to be resorted to, it will be well to provide two or more ordinary hand pumps of the bilge or diaphragm type, with 3½ in. suction, and 3 ten-foot lengths of heavy rope-wound hose and couplings for each pump, together with good foot-valves and strainers for each. Sometimes the results of an examination are lost for the lack of such simple equipment.

Prospecting by Shaft Sinking.—I will dismiss with a few words the subject of prospecting by shafts, as this desirable method is rarely possible. The ground is laid out by survey, and shafts are sunk at intervals of 100 to 200 ft. in lines across the valley or gravel bench. The shafts should preferably be 4½-ft. square, and as little timber used as possible. A cut in each of the four sides of the shaft is made as each foot in depth is attained, and this gravel is hoisted and piled separately. The rocker is set up, and the panner will, after determining the weight and cubic contents of a bucket of gravel used for measuring, put the representative sample from each foot or two feet through the rocker, and clean up. The number of colors and sizes for each unit of depth will be entered in the shaft log, and then the gold placed in the concentrate cup or bowl. At the end a few drops of quicksilver will collect all the gold from the test, and the amalgam is placed in a bottle to be parted in the evening with nitric acid and the gold weighed. A factor on the safe side is the fact that the recovered gold weighed after parting is nearly pure, but is still valued as placer gold. When colors are coarse, amalgamation is not necessary, as the cleaning may be done with a magnet and the non-magnetic residue blown away.

Shaft sinking varies in cost from $1 to as high as $5 per foot. The price and quality of labor, amount of timbering and pumping necessary all vary with each separate case, so for the cost and rate of sinking of gravel shafts no rule can be laid down. If values are found to be uniform in a number of preliminary shafts, the subsequent ones may be spaced one to an acre or one to each 5 or 10 acres, according to the degree of uniformity with which it is found that the gold is disseminated.

Shaft sinking has the advantage of affording large samples, generally from 500 to 1,000 lb., giving a good idea of the amount of clay, size of boulders, etc., and the character of the bedrock. Unfortunately, the ground easiest for prospect by shafts is generally unfavorable for working on account of lack of water, presence of clay or heavy depth of soil over-burden. In loose gravel, the most favorable for placer mining by any form, shaft sinking is rarely possible.

Type of Drill.—The engineer is frequently unable to determine beforehand what method of testing he will be obliged to use. In such a case, it is essential that at least one drill should be sent to the property. The type of drill will be determined largely by conditions. As regards cost, it may be taken as an average that the heavy power drill with 100 ft. of six-inch casing will cost about $3,000 (£600) landed on a property in Alaska, Siberia, South America, or Africa, while the ordinary type of hand drill with 50 ft. of 4-in. casing will cost about one-third this amount. With the power drill it is absolutely essential that an expert machine man should be secured in advance, either locally, or if this is not possible, he must be engaged from some field where placer testing by power drills is practiced. It is not advisable to engage an oil driller or well driller for placer drilling. Keystone drillers get $3.50 per shift in California, and from $5 to $6 and board in Alaska. The average expense of such a man sent to an isolated field will be $7 per day plus his expenses there and return. This is one of the items which indicates that the sampling of placer properties is not a cheap process in any country. Firemen and even assistant machine-men for operating the drill on night shift may generally be locally obtained.

Hand drills, such as those of the Banca and Empire type, require no skilled labor, and a drill crew of local labor can be taught in a few hours to satisfactorily operate the drill.

Panners or panmen who can be trusted can generally be obtained in placer districts such as California, New Zealand, or Alaska. Great difficulty is experienced in getting satisfactory panners in some regions and this adds seriously to the expense of examinations in those countries as comparatively highly paid assistants must be taken with the expedition. The engineer in charge will always do as much of the panning of samples himself as possible, but on examinations of any size he is otherwise employed.

In the older regions, like the dredging fields of New Zealand or California, the type of drill in local use will generally give the most satisfaction, because trained drill crews may be engaged before the engineer leaves his home office to take the work on contract at stipulated rates. Thus, in Nome, Alaska, drill crews operating power drills with wide tires, especially adapted for the tundra, will contract for drilling in frozen ground at 80c per foot, and in unfrozen ground for $1.50 per foot. This it can be easily seen is an immense advantage, as the engineer is not obliged to purchase an expensive drill at a distance, have it shipped to Nome, and run the risk of losing a portion of his time through its non-arrival.

At Oroville, California, or in the Yuba or Folsom fields, drill crews may be contracted, at so much per foot of hole, the examining engineer being able to give all his attention to the washing of the samples, watching only to see that the drilling is done in such a manner that the correctness of the sample is not interfered with.

In isolated parts of South America, Africa, or Siberia, on the other hand, where such contract crews for gravel work do not exist, and where roads for transport are generally lacking, it will generally be found expedient to use the hand type of drill, if possible. An exception is in the deep gravel areas of the Lena and Trans-Baikal regions of Siberia, where, notwithstanding the difficulties of transport, it is in most cases advisable to use the power drill.

In nearly all cases, however, of an extensive examination of gravel deposits, the hand drill will be found a valuable accessory. No extensive gravel examination should be undertaken without such a machine as a part of the equipment. The hand drill is easy to transport, its total weight, including 50 ft. of casing, rods and necessary tools, not exceeding 2500 lb., while the heaviest parts weigh less than 75 lb. each. It is quickly assembled in the field, and common labor in any country can be soon taught its mechanical features. Such drills have been successfully used not only in English-speaking countries, but in the Malay States, South America, Africa, Siberia, and elsewhere, using native labor. I have seen such drills transported on mule back and even on reindeer.

Drilling with Power Drills.—For the use of the Keystone or other power drill the ground is first laid out in squares by co-ordinates from one acre to in some cases 10 acres, and a post to mark a proposed drill hole is placed at each corner. The drill, which is of the 6-in. type, weighs with its tools about eight tons, and consists of a carriage with wide wheels, on which is mounted an 8-horsepower steam engine and boiler or liquid fuel motor, a detachable derrick, a walking beam arrangement, and a mechanism for adjusting the walking beam. In the traction type, the power may be used when advisable to move the drill from one hole to another. This in turn gives a vertical reciprocating motion with a 30-in. stroke to a 1¾-in. cable passing over a sheave at the top of the derrick. The rope passes down the front and centre of the derrick frame and is attached by means of a rope-socket and a heavy 4-in. stem, 12 ft. long, to a long thin bladed drill bit. A tool called the jars is also occasionally inserted between the rope socket and the stem. The string of tools so-called weighs from 700 to 1,200 lb. according to the number and size used. A hole is dug with pick and shovel and a five-foot length of casing of 6-in. *inside* diameter pipe fitted at its lower end with a drive shoe of which the cutting edge is 7½ in. *outside* diameter, is set in the hole, plumbed and the dirt filled in around it. Next, if drilling is necessary, the drill bit is lowered through the casing and a few inches are drilled below the casing. If the ground is soft a driving cup is screwed to the pipe and the casing is driven for say two to six feet through the top soil or sand. This description of driving and pulling the casing is quoted from a paper by N. B. Knox*

"The driving is accomplished by striking the driving head with a couple of iron blocks clamped to the stem by means of two 1¾-in. bolts, the weight of the string of tools acting as a hammer. After driving, the driving blocks are removed when the first length of casing is driven down to head, the driving cap is removed, a second section of casing is screwed on the first, the driving cap replaced, and drilling resumed. When the required depth is reached, determined either by striking bedrock or passing through the pay stratum, the hole may be considered finished, and the next step is to pull up the casing. This is accomplished by removing the bit, stem and jars, and replacing them by what is known as the pulling or pipe jars. These consist of an iron boss fixed to the end of a rod 4½ ft. long. Above the boss is a 1-in. thick plate—the 'knocking head'—provided with threads which are screwed into the sleeve of the top section of casing. The stem of the boss passes through a square hole in the plate. The walking beam is set into motion, and the string of casing is raised by the boss striking against the knocking head. As each section of casing is raised,

*Trans. Inst. Min. and Met., Vol. 12; 1902.

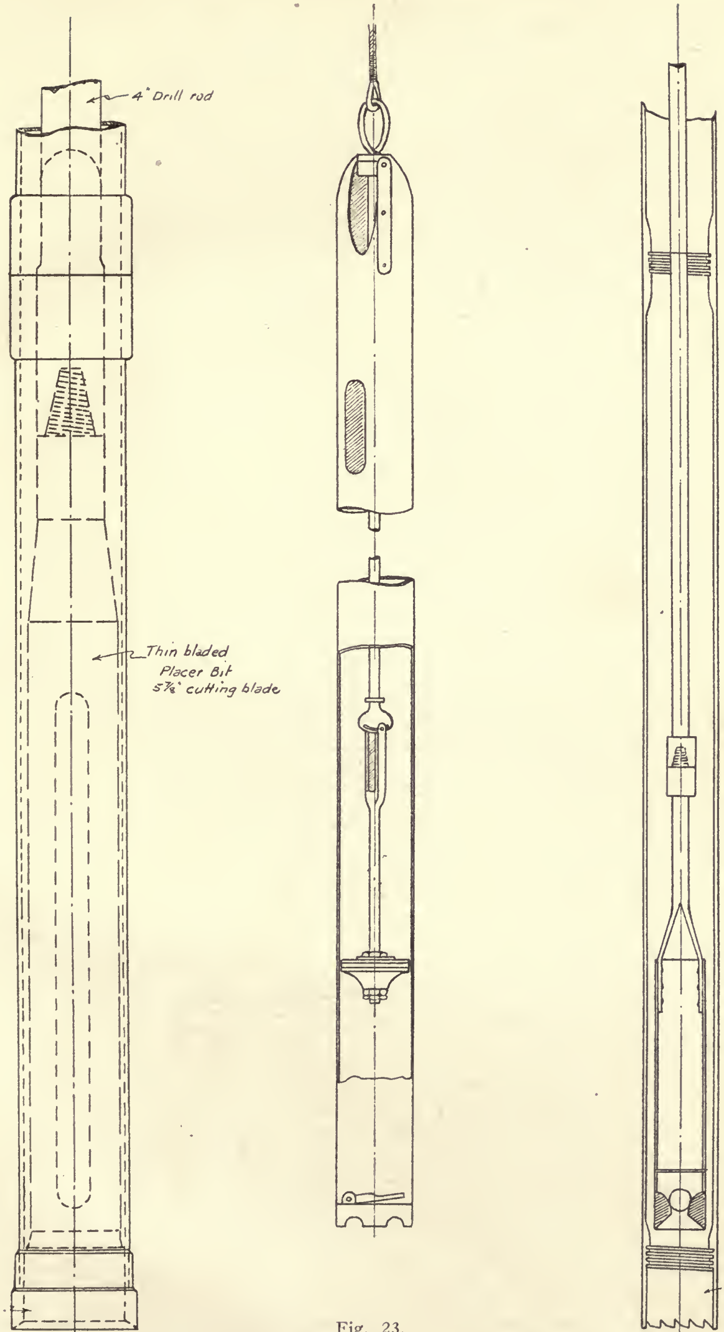

Fig. 23.

CASING AND DRILLING BIT FOR 6" POWER DRILL WITH DRIVEN CASING.

KEYSTONE 8 FT. X 4½-IN. SAND-PUMP.

4" HAND DRILL WITH ROTARY CASING, SHOWING SAND-PUMP AND DRILL ROD.

it is unscrewed, and the knocking plate screwed on the next. If care is used in keeping the threads of the casing clean, the casings can be used for a long time. It is rarely that a casing is lost."

Great care is necessary that the drill bit is never allowed to go below the casing, but from 12 to 24 inches of the core must be allowed to remain in the pipe. The sand-pump, which is attached to a separate small line and operated by a separate reel, is lowered into the hole every foot drilled, and the drillings pumped out within 6 inches of the bottom of the core, the pump is lifted and its contents dumped into a box about 8 feet long and 12 in. wide. The depth of core is carefully measured each time by a system of marking the drill stem and comparing the relative depths below surface of the top of the core and the length of the casing.

Each foot of the core is panned and the slimes are afterwards treated in the rocker. The gold from each panning is estimated and classified as to number and size of colors, and the appropriate entry made in the drill log (Fig. 24). After bedrock is reached and the hole finished, the slime from all the pannings is rocked and the fine gold recovered is added to the samples. All the gold is then amalgamated with a globule of quicksilver, the amalgam placed in a bottle and afterward freed with acid from the quicksilver and weighed. The details of entries in the drill log and the time log (Fig. 25) shown here, which are for hand drill work, indicate the depth of hole, character of material, bedrock, whether the drill bit gets below the casing, so as to endanger salting the result, but also an important factor, at what depth the pay-layers are found, and the horizons at which the gold is concentrated. From one to two feet are drilled in bedrock even after no more colors are found, for the sake of safety. Water is always kept in the hole artificially if it does not flow into the hole naturally from the ground.

Good Keystone work runs from 20 to 30 ft. of drilling in 12 hours, and the cost averages in standard California practice, including the services of the panner, \$2 per foot drilled. In Alaska in the summer of 1912, in solidly frozen ground, where casing was almost entirely dispensed with, 2,500 ft. of drilling was done with a heavy specially designed power drill with 6½-in. bit, at a cost of approximately \$1.15 per foot. About 40 ft. per shift was made, with a fuel consumption of about half a cord of wood, and 1,000 gallons of water for drilling and washing samples.

Calculation of Value.—The value of the ground is calculated as follows: The cylinder of material represented by the core is 7½ in. in diameter multiplied by the depth, if the drilling has been so carefully done as to allow no gravel to run into the pipe. Each foot of the core contains 0.3 cu. ft. or $^1/_{90}$ of a cu. yd. In the case of a hole 40 ft. deep from which 12c. in actual gold was recovered there would be $\frac{40}{90}$ cu. yd. worth 12c., or $\frac{12 \times 90}{40} = 27.2$c. per cu. yd. In this case the factor used is 0.3. It has been found by elaborate experiments that this factor is somewhat too large, and 0.27 is the one

DRILL LOG

Date July 1-1912

Hole No. 6-line E. Crew No. .... Tract Phoenix

Started Hole .... Finished ....

Depths IN FT AND IN.			Cores IN INCHES				Colors			Firmness				
Depth represented	Depth of pipe	Depth drilled below pipe	Core after driving	Core for depth driven	Core after pumping	No. of pans	1	2	3	Driving	Drilling	~~Lumps (clay)~~ Drove	~~Slime~~ Rotated	Formation
	2-6		not	Panned								Excavated		M. G.
	5		18	18	0		-	-	-	Easy	Easy			F.G.-Sand
	6		9	9	0		-	-	-	"	"			do.
	7		10	10	0		-	-	-	"	"			do.
	7-6		-	-	-		-	-	-	-	"			do.
	9		15	15	0		-	-	-	Easy	"			do.
	9-6		-	-	-		Trace			-	"			do.
	10		6	6	0					Easy	"			do
	11		13	13	0			do		"	Med			Sand-F.G.
	11-13		-	-	-			do		-	"			do
	12		12	12	0					Easy	Hard			do.
	13		12	12	0		-	-	-	Med	Med.			F.G-Sand.
	14		12	12	0		-	-	-	"	"			do
	15		8	8	0		-	-	-	Easy	"			Sand-Silt
	16		12	12	0		-	4	2	Med	"			F.G.Sand-Silt
	17		11	11	0					"	"			Sand-Clay Soft Schist
	17-6		6	6	0		4	-	1/2	Hard	"			Soft Schist and Clay
	18		6	6	0					"	"			do
	18-6		8	8	0		-	-	1	"	"			do
	19		8	8	0					"	"			do
	20		11	11	0		-	-	1	"	"			M-H-Fractured Schist & Clay.
	20-6		9	9	0		-	-	-	"	Hard			do
	21		7	7	0		-	-	-	"	"			do

Total Depth 21 ft.

Depth to Bedrock 17 ft.

Depth to Water Level 1 1/4 ft.

Measured Volume 1.37 cu. ft (for last 18 ft)

John Smith Panman

Fig. 24. DRILL LOG.

TIME LOG

Date July 1-1912

Hole No. 6 line E Crew No. 1 Tract Phoenix

Started Hole 12.35 p.m. July 1-12 Finished 11.50 a.m. July 2-1912

	Time of day particular operation is finished	Nature of work	Hr. Min. Consumed in							Remarks
			Drilling operations	Oiling Putting on pipe	Pulling pipe	Moving	Setting up	Delays	Depth of pipe	
July 1	P.M. 12.35	Began work								
	12.55	Moving				20				
	1.20	Clearing					25			
	1.35	Excavating	15						2½	
	2.05	Drilling	30						5	
	2.10	Pipe		5						
	2.18	Drilling	8						6	
	2.26	"	8						7	
	2.35	"	9						8	
	2.45	"	10						9	
	3.00	"	15						10	
	3.05	Pipe		5						
	3.23	Drilling	18						11	
	3.45	"	22						12	
	4.00	"	15						13	
	4.14	"	14						14	
	4.25	"	11						15	
	4.31	Pipe		6						
	5.00	Drilling	29						16	
Jul. 2.	A.M. 8.40	Began work								
	9.05	Drilling	25						17	
	9.30	"	25						18	
	10.04	"	34						19	
	10.32	"	28						20	
	10.40	Pipe		8						
	11.25	Drilling	45						21	
	11.50	Pulling pipe			25					
	Totals		6.01	0:24	0:25	0:20	0:25			7 hrs 35 min

John Smith Panman

Fig. 25. TIME LOG.

ordinarily used. This means that a cylinder 100 ft. long equals 1 cu. yd. In the illustration just given, the result would be 30c. per cu. yd., using 0.27 as a factor.

Formula.

Let C = money value of sample
D = depth in feet
V = value per cubic yard
L = length of cylinder which will equal 1 cubic yard

Then $$\frac{C \times L}{D} = V$$

or substituting the figures above quoted we have:

$$\frac{12 \times 100}{40} = 30c.$$

Looking now at the 'Prospecting Report Sheet' (Fig. 26) on which the final results for each hole are calculated, the data necessary in calculating the final results are shown under the heads *'Average value per cubic yard'* and *'Depth used in calculation.'* These two multiplied together give the number of foot-cents for each hole. The foot-cents added together and divided by the number of feet drilled give the average value per cubic yard for the gravel of a given line of holes, or a given block of ground, providing the holes are equally spaced, while the average depth in feet is the total footage divided by the number of holes. Erratic results on the high side are neglected, and where the values gradually decrease on both sides of a large tract, the engineer will finally draw lines on the completed drilling plan, which define an area of pay ground. If this pay-channel has sufficient width and length to afford a yardage for a reasonable life for the extent of equipment in dredges or other apparatus which is practicable, and yield the desired margin of profit, the property may be recommended.

Drilling with Hand Drills.—Where depths of material from surface to bedrock do not exceed 40 ft., the hand drill may be used. There are three types of these drills in common use, two which are practically of the same design and known respectively as the Banca and the Empire, and a third, which differs in design and mode of operation, is known as the Dimond. They all accomplish practically the same result, in the same time, and at the same cost.

In isolated localities, devoid of roads, and where the time factor is important, the hand drill has many things in its favor. I do not recommend its use where it can be avoided, for the following reasons:

1. Instead of six-inch casing, only four-inch is used, whereby a smaller sample is obtained.

2. It has not the power to penetrate heavily frozen ground or large boulders.

3. The type of sand-pump used is of the ball type rather than the clap valve, and the ground-up material brought to the surface is not as certain to represent the exact portion of the hole as in the case of the power drill.

4. There is a greater opportunity for tampering with the sample, as four men stand continually on the platform in drilling, and although there is greater danger from salting in shaft-sinking, this danger does not exist to any extent in the use of power drills.

PROSPECTING REPORT

Examination Phoenix Date July 1 - 1912

No. of Hole 6 line "E" No. on Map Crew No. 1

Location of Hole 100 ft to left of nos, on line with 1. to 5.

Method of Prospecting by Hand drilling Machine Used Empire Drill

Started Hole 12.35 p.m. July 1/12 Finished Hole 11.50 a.m. July 2/12 Days Worked 7 hr, 35 min

Depth of Top Soil —

Depth to Bedrock 17 ft.

Depth Drilled 21 feet.

Depth Used in Calculations 20 feet.

Depth to Water Level 1 1/4 feet.

Distribution of Gold 10, 11, 12, 16, 17, 18, 19, 20 feet.

Nature of Top Soil no top soil

Nature of Gravel Loose to medium tight, angular and sub-angular, fine.

Nature of Gold Bearing Strata Fine gravel and sand, clay, slime and bedrock.

Nature of Bedrock Soft to medium hard fractured schist and clay.

Time Consumed in—Moving 20. m. Setting Up 25 m. Drilling Operations 6 hr. 25 min

Pulling Pipe 25 m. Delays — Total 7 hr. 35 min.

Causes of Delays

Mg. of Gold Recovered 11

Factors Adopted ~~Fineness~~ Value of Gold .06 ¢

Cu. Ft. per Ft. Depth of Hole .1134 Average Value per Cubic Yard 7.857¢

Remarks

Signed John Smith, panner

Checked by

Approved by

Fig. 26. PROSPECTING REPORT.

The operation of the hand 4-in. drill with rotated casing is as follows: The hole is started with an auger or, if necessary, by digging for 18 inches. In the case of soft peat or soil over-burden several feet may be made with the auger.

A 5-ft. length of 4-in. casing with a 0.38-ft. diameter cutting shoe screwed to the bottom is set in the auger hole and plumbed, and the heavy platform is screwed on top and four men mount on it. The men, platform and casing will weigh from 1,000 to 1,500 lb., according to the depth attained and amount of casing used. A heavy log is now used as a ram to drive the casing for two or three feet into the ground. For medium ground this driving process is not much employed after starting the hole. It is made use of in tight or coarse gravel, in which case also a drilling bit is used with the string of rods as well as the drilling pump. The ordinary procedure is to rotate the pipe by means of long rods or a sweep propelled in a circle by a horse, the weighted platform causing the casing to sink. A string of rods, graduated in inches for measurement, and with the drilling pump screwed to the bottom, is let down the hole, and drilling, pumping, and rotating go on all at the same time. This combined work is an advantage of this type of drill, not possible with the power drill. The same care is taken as in Keystone drilling never to let the pump or bit get below the casing, and there is no essential difference in the manner of treating the material as it is drawn up each time by the pump, dumped into the clean-up box and panned, except that smaller amounts are dealt with.

When the casing is driven to bedrock and the hole finished, it is pulled by means of a clamp and long lever acting on a frame bolted together of three pieces of structural steel.

The system of keeping the notes is practically the same as with the power drill, and the sheets here exhibited (Figs. 24, 25, 26) are in fact records of Empire hand-drill work. The rate of progress per day varies considerably. It is safe to put it at 15 to 20 ft. per 10-hour day counting all time consumed. The cost of drilling, including salary of the assistant or panner in charge, will range from 50c. to $1.15 per foot. Counting labor alone it will be from 25c. to 40c. per foot. A chisel bit is used to cut through small boulders, but much time is lost when large boulders are encountered, for it is useless to try to drill through them, and if they exist in large numbers, hand drilling is generally labor lost; otherwise the drill will successfully cope with a surprising number of difficulties.

In calculating the value of the gravel instead of 0.27 the pipe-factor used is 0.1134. This is the volume of 1 foot in length of core, which is a cylinder 0.38 ft. in diameter, the size of the cutting shoe.

It takes 238.1 ft. of core to make 1 cu. yd. instead of 100 ft., as with the 6-in. power drill. Thus if the total gold found in the hole amounts to 6c. and the depth 30 ft., then applying the formula we have:

$$\frac{C \times L}{D} = V$$

or substituting $\frac{6 \times 238.1}{30} = 47.6$c. per cu. yd.

When the value has been determined the averaging of the holes is figured as previously described.

A few test shafts should be sunk, if possible, around the drill holes which have been finished, to test the accuracy of the results. Unfortunately, this is seldom possible on account of water. It may be said, however, that the drilling of placer gravels, both with hand and power drills, has long passed the experimental stage. With the allowance of the safety factor, which every engineer will make for difficulties attending recovery of the prospected values in the subsequent mining, it is doubtful if drilling can be superseded by any system of prospecting alluvial gravels which is more rapid, economical or generally satisfactory.

INDEX

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MAR 18 1956 LU

1Mar'57BR

REC'D LD
FEB 15 1957

JUN 4 1957 S

REC'D LD
MAY 31 1957

4May'61DA

Sann Maung

REC'D LD
MAY 29 1961

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